I. C. 6710

May, 1933

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

EXPLOSIONS IN PENNSYLVANIA COAL MINES, 1870-1932

BY

J. J. FORBES AND H. B. HUMPHREY

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By J. J. Forbes[2] and H. B. Humphrey[3]

PURPOSE OF THIS REPORT

In accident prevention, a study of statistical data is necessary to determine primary causes of accidents to devise means to eliminate causes at the original sources. Explosions in coal mines of the United States during the 10 years ending December 31, 1931, ranked third in the causes of fatal accidents; they constituted 16.2 per cent of all fatal accidents in bituminous mines and 8.4 per cent of all fatal accidents in anthracite mines. A study of coal-mine explosions in Pennsylvania, with approximately 160,000 men employed in anthracite mines and about 145,000 in bituminous mines or about ~~2~~305,000 out of the approximately ~~525~~,000 men employed in all coal mines of the United States, should give information of value to all persons endeavoring to reduce or eliminate coal-mine explosions. This circular reviews explosions of gas or dust or of both in the coal mines of Pennsylvania to show the hazard of gas and dust, to point out the influence of certain factors in causing explosions, and to emphasize the efficacy of explosion-preventive measures.

The data published annually by the State Department of Mines and of the United States Bureau of Mines were used in compiling the tables of explosions. Through the cooperation of W. H. Glasgow, secretary, Department of Mines of Pennsylvania, and his assistants, many doubtful cases were clarified, so that as far as recorded the history is comparatively complete. Figures used in compiling some of the tables were furnished by W. W. Adams, chief statistician, demographical division, United States Bureau of Mines.

EXPLOSIONS IN ANTHRACITE MINES

Anthracite in the Wyoming Basin of Pennsylvania was used locally by gunsmiths in 1755, by blacksmiths in 1769, and as domestic fuel in 1808.

1 The Bureau of Mines will welcome the reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6710."

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Mining and shipments developed slowly between 1806 and 1812, but more rapidly after that.

Early mining along the outcrops gave essentially no trouble from gas, but by 1870, when State inspection was established, explosions were frequent; ventilation was crude and accumulations of gas were generally "brushed" or burned out.

Table 1 lists 1,232 ignitions and explosions of gas in anthracite mines from 1870 to 1932, inclusive, every one of which was accompanied by loss of life. Some details are given of 110 representative cases in which five or more lives were lost or in which it appeared that some violence was present.

Fatalities[4]

During the period 1870 to 1932, inclusive, gas explosions caused 2,213 fatalities, or an average of about two deaths for each ignition or explosion. Records are not available as to the number injured in these and in other explosions where no deaths occurred, but from a study of explosion reports it appears probable that the number injured exceeds the number killed. In the 110 cases described, 599 were killed and 294 were injured. As shown in Table 2, fatalities from gas explosions averaged 8.3 percent of the total fatalities in Pennsylvania anthracite mines from 1870 to 1931, varying from 6.0 percent in 1931 to 13.1 percent in 1927.

Table 2. - Fatalities from gas explosions in anthracite mines

Period, years	Fatalities per million		Percent of total number killed
	Tons	Man-hours (surface and underground)	
1870	1.09	0.24	8.1
1871 - 1880	1.11	.23	12.3
1881 - 1890	.66	.13	7.9
1891 - 1900	.71	.14	9.8
1901 - 1910	.47	.11	6.2
1911 - 1920	.47	.12	7.3
1921 - 1925	.58	.15	10.0
1926	.68	.18	12.8
1927	.80	.22	13.1
1928	.42	.12	7.1
1929	.42	.11	6.4
1930	.59	.16	9.3
1931	.39	.11	6.0
Total	0.59	0.14	8.3

4 The number of cases and men killed differ from yearly totals given by the Bureau of Mines and the State due to the inclusion of fatalities segregated by them under other headings and some in mines not under State control.

Table 1.- Gas explosions in anthracite mines of Pennsylvania

Case	Date	County	Type of opening	Tons per day	Men employed	Men in mine	Extent of explosion	Cause of accumulation	Source of ignition	Other factors	Possible means of prevention at time	Killed	Injured	Escaped from area uninjured	Other cases: Ignitions	Other cases: Killed	Total yearly gas fatalities
1	1847 Feb. 19	Schuylkill	-	-	-	-	Small	Poor ventilation	Open light	Testing air with lamps	Ventilation, flame safety lamps	7	-	-	-	-	7
2	1866 Aug. 26	do.	-	-	-	-	Small		Open light		Flame safety lamps	3	-	-	-	-	-
3	1866 Summer	Luzerne	Shaft	-	-	3	Violent	Ventilation interrupted by shaft repairs	Furnace at shaft bottom		Continuous ventilation	3	-	-	-	-	-
4	1866 Autumn	do.	do.	-	10	10	do.	Fall released gas	Open light	Had finished repairs to shaft	Safety lamps	-	9	1	-	-	6
5	1870 Aug. 31	Carbon	-	200	250	-	Small	Gas commonly present	Unlocked flame safety lamp	Miner removed gauze from flame safety lamp	Ventilation and proper use of lamps	2	-	-	9	15	17
6	1871 June 2	Schuylkill	Slope	150	110	-	Small	Insufficient ventilation	Open light		Ventilation and inspection, flame safety lamps	3	3	-	-	-	-
7	1871 Aug. 14	Luzerne	Shaft	30	25	20	-		do.		Proper ventilation and flame safety lamps	17	-	3	-	-	-
8	1871 Oct. 2	Schuylkill	Slope	150	100	-	Small	Released by blasting	do.	Fall forced gas over lights	do.	5	1	-	11	19	44
9	1872 Feb. 8	Luzerne	Tunnel and shaft	-	40	5	Violent	Poor ventilation and open door	do.	Reopening and repairing; no inspection before men entered	Inspection and flame safety lamps	4	1	-	17	27	31
10	1873 June 10	Northumberland	Drift and slope	75	50	-	Small	Unventilated old workings	Fire boss exploring with open light	9 men on return air killed by carbon monoxide	Ventilation and flame safety lamps	10	-	-	9	18	28
	1874* 1/	-	-	-	-	-	-		-	-	-	-	-	-	15	33	33
11	1875 Dec. 28	Luzerne	Shaft	300	220	-	Local	Broke into old workings	Open light	Men went in from nongassy mine	Instruction, supervision, safety lamps	3	4	-	9	16	19
12	1876 Feb. 12	do.	do.	750	240	10	do.	Ventilation repairs on idle day	do.	Man went off intake airway	Supervision and flame safety lamps	4	2	4	-	-	-
13	1876 Apr. 12	Carbon	Shaft and slope	550	320	-	do.	Ventilation, pipes changed in a heading	do.	Men went in against orders	Inspection and flame safety lamps	4	4	1	11	21	29
14	1877 May 9	Schuylkill	Shaft	850	570	-	Small	Removal of door	do.	Fall forced gas over lights	do.	7	6	4	5	9	16
15	1878 Jan. 15	do.	Slope	500	300	-	Local	Outburst of gas	Davy flame safety lamp	Blew on flame and flame came through gauze	Proper use of lamps	5	-	-	-	-	-
16	1878 Oct. 8	Luzerne	do.	900	450	-	do.	Feeders	Fire	Rekindling of fire	Precaution in fighting fires	4	-	1	8	14	23
17	1879 Jan. 9	do.	Shaft	400	160	3	do.	High concentration in return airway	Open light	Fire boss entered without testing air	Use of safety lamps	1	1	-	-	-	-
18	1879 May 6	Carbon	do.	950	600	20	do.	Feeders in test hole	Fire	Extinguished flames, but coals reignited gas	Possibly sealing fires	6	2	-	-	-	-
19	1879	Luzerne	do.	1,400	820	-	Violent	Cavity in sagging roof	Open light	Gas released suddenly	Flame safety lamps	-	4	-	-	-	-
20	1879 Nov. 2	do.	Slope	900	445	5	Local	Cave-in	do.	Timber crew went in Sunday morning; no inspection made	Flame safety lamps and inspection	5	-	-	9	17	29
21	1880 Mar. 5	do.	do.	1,200	725	-	Small	Unventilated idle workings	do.	Fire boss and party exploring; no tests for gas	do.	6	-	-	-	-	-
22	1880 May 3	Northumberland	do.	550	400	-	do.	Gas released by run of coal	do.	Extracting pillars	Supervision, inspection, and flame safety lamps	5	-	-	6	10	21
	1881*	-	-	-	-	-	-		-	-	-	-	-	-	12	27	27
23	1882 May 24	Schuylkill	do.	700	445	-	Violent	Squeeze falls blocked airway	do.	Open light in safety-lamp section	Supervision, no open lights	5	2	-	-	-	-
24	1882 June 15	Luzerne	Shaft	-	100	6	Local	Sinking shaft; pumps stopped	do.	Ventilation by exhaust of pumps only	Continuous ventilation and flame safety lamps	3	3	-	10	17	25
	1883*	-	-	-	-	-	-	-	-	-	-	-	-	-	15	28	28
25	1884 Dec. 8	Northumberland	Slope	220	170	-	Small	Cut into gas pocket	Broken flame safety lamp	Sudden inrush of gas	Care in use of flame safety lamp	4	1	2	5	8	12
26	1885 Feb. 17	Luzerne	Shaft	350	240	-	Local	Fan stopped 20 minutes with hot bearing	Open light	Had sent in to call men out	Second or stand-by fan, flame safety lamps	2	11	-	-	-	-
27	1885 Oct. 21	do.	do.	500	380	-	do.	Reopening after being idle 9 months	do.	One gas pocket burned, second exploded	Inspection, ventilation, flame safety lamps	6	10	-	6	15	23
28	1886 Aug. 30	Lackawanna	Slope	300	180	7	do.	Fan stopped 1 hour, and fall on slope	do.	Fire boss and 6 men went in with open lights	Inspection and flame safety lamps	6	1	-	-	-	-
29	1886 Nov. 26	Luzerne	Shaft	500	260	52	do.	Pumps stopped, sump full	do.	Man went to sump; passed danger board	Supervision, flame safety lamps	12	20	15	5	5	23
30	1887 Mar. 30	Lackawanna	Slope and shaft	1,200	600	-	Small	Poor ventilation and inspection	do.	Gas moved over lights	Ventilation, inspection, flame safety lamps	2	5	-	-	-	-
31	1887 June 23	Luzerne	Shaft	2,000	1,000	-	do.	do.	do.	Went into uninspected working	Inspection and flame safety lamps	3	2	-	8	10	15
32	1888 Jan. 27	do.	do.	2,000	1,000	-	do.	Open door	do.	Gas carried over mason's lights	Flame safety lamps and continuous ventilation	1	3	-	12	19	20
	1889*	-	-	-	-	-	-	-	-	-	-	-	-	-	18	29	29
33	1890 Feb. 1	do.	do.	2,000	1,000	-	Local	Cave and squeeze; repair work only	do.	No inspection, carried open lights	Inspection and flame safety lamps	8	5	-	-	-	-
34	1890 Apr. 2	do.	Drift and slope	2,000	1,000	-	Small	Unventilated face of breast	do.	In moving gas let it go to next face	Not move gas with men in mine; safety lamps	5	4	-	-	-	-
35	1890 May 15	do.	do.	550	375	-	Strong	Cave-in blocked airway	do.	Man tried to light lamp while getting out	Prohibit matches; remove men in time	26	2	-	-	-	-
36	1890 Aug. 20	do.	Slope	550	280	4	Violent	Surface repairs; mine idle for month	do.	Incomplete inspection; officials entered	Inspection and ventilation, flame safety lamps	4	0	0	-	-	-
37	1890 Oct. 8	do.	Shaft and slope	1,500	600	-	Strong	Idle breast, unventilated	Blasting (dynamite)	Blast broke through into idle breast	Ventilate all open workings	2	1	-	10	15	60

1/ Years marked with an asterisk are those in which there were no major accidents, or in which no accidents occurred.

Table 1.- Gas explosions in anthracite mines of Pennsylvania - Continued

Case	Date	County	Type of opening	Tons per day	Men employed	Men in mine	Extent of explosion	Cause of accumulation	Source of ignition	Other factors	Possible means of prevention at time	Killed	Injured	Escaped from area uninjured	Other cases: Ignitions	Other cases: Killed	Total yearly gas fatalities
38	1891 Nov. 8	Luzerne	Shaft	1,900	1,600	15	Strong	Changing ventilation, idle days	Davy lamp	Gas came through "missed" crosscut	Supervision	12	2	1	-	-	-
39	1891 Dec. 31	do.	Slope	750	375	-	Local	Repairing ventilation in idle section	do.	Strong current pulled flame through gauze	Supervision; stay on intake	2	1	0	15	25	39
40	1892 Mar. 9	do.	Shaft	1,200	650	-	do.	Air hose removed while blasting	Blasting (dynamite)	Shot two holes with battery	Ventilation	3	2	-	-	-	-
41	1892 May 31	do.	Slope	750	400	-	do.	Insufficient ventilation	Open light	Went in ahead of fire boss inspection	Inspection, flame safety lamp	4	7	-	-	-	-
42	1892 July 23	Schuylkill	do.	200	400	-	Strong	Gas freed by blast	Davy lamp	Lamp left in gas when men ran	More care, extinguish lights	15	-	-	-	-	-
43	1892 Nov. 9	Luzerne	Shaft	1,000	500	-	do.	Poorly regulated ventilation being fixed	Open light	Erecting brattice, forced gas over open light	Only competent men and flame safety lamps to clear gas with others out of mine	2	3	-	27	34	58
44	1893 June 22	do.	do.	2,000	1,100	-	Violent	Driver left door open	do.	Closing door moved gas onto gangway	Double doors, flame safety lamps	5	2	-	-	-	-
45	1893 Sept. 21	do.	Slope	1,400	650	-	Strong	Changing air	do.	Ignited by foreman	Inspection, flame safety lamps	6	2	2	19	35	46
	1894* 1/	-	-	-	-	-	-	-	-	-	-	-	-	-	15	29	29
46	1895 Feb. 18	Schuylkill	do.	1,300	750	-	do.	Released by run of coal	Davy lamp	Strong current pulled flame through gauze	Extinguish lamps in time	5	0	0	-	-	-
47	1895 Oct. 7	Luzerne	do.	1,000	530	-	do.	Old working not examined	Open light	Fire boss and party of young engineers	Proper inspection, flame safety lamps	7	0	3	13	19	31
48	1896 Sept. 24	Schuylkill	Shaft	300	190	-	Great violence	Gas burst released by blast	Davy lamp	Flame forced through gauze	Blast only after men are off return side and some distance away	4	1	-	-	-	-
49	1896 Oct. 29	Luzerne	do.	2,600	800	9	Violent	Temporary change in ventilation, idle day	Blasting (50% dynamite and electric battery)	Blasting for overcast; ventilation disturbed	Inspection before blasting	6	-	3	18	32	42
	1897*	-	-	-	-	-	-	-	-	-	-	-	-	-	21	36	36
50	1898 Oct. 13	Carbon	Slope	1,200	260	13	Small	Sealed area	Fire	Filling behind seals with water	Remove men when seals are in	4	7	2	19	36	40
	1899*	-	-	-	-	-	-	-	-	-	-	-	-	-	16	28	28
51	1900 Nov. 9	Schuylkill	do.	1,000	375	-	Strong	Temporary change, building overcast	Open light	Miner went into abandoned section	Ventilate all workings, flame safety lamps	7	8	-	20	31	39
52	1901 Oct. 25	Luzerne	do.	2,000	880	-	Local	Feeders lit by open light but extinguished	Fire	Coal embers reignited explosive mixture	Flame safety lamps, care with fires	6	6	-	14	27	33
53	1902 Nov. 29	Northumberland	Shaft	1,400	800	-	do.	Poorly ventilated faces	Blasting (dynamite)	Gas known to be present	Ventilation before blasting	7	5	-	9	13	20
	1903*	-	-	-	-	-	-	-	-	-	-	-	-	-	15	26	26
	1904*	-	-	-	-	-	-	-	-	-	-	-	-	-	20	30	30
54	1905 Dec. 13	do.	do.	1,100	800	-	do.	Large burst of gas	Smoking	Locked safety lamps used	No smoking	4	2	-	29	30	34
55	1906 May 14	Luzerne	Slope	1,100	590	-	Violent	Fall on gangway	Open light	Man entered workings reported clear	True reports, flame safety lamps	2	2	-	-	-	-
56	1906 Aug. 6	do.	Shaft	1,800	1,100	-	Strong	Poor ventilation in face	Blasting (dynamite)	Electric magneto firing unit	Permissible blasting units	6	1	-	30	35	43
57	1907 Mar. 2	Lackawanna	do.	1,200	490	-	Local	Open doors	Open light	"Mine well ventilated"	Eliminate or safeguard doors; flame safety lamps	7	3	-	-	-	-
58	1907 June 18	do.	do.	2,000	1,200	-	Strong	Stoppings blown out by first small explosion	do.	Men went back and set off second strong explosion	Supervision, flame safety lamps	7	2	3	22	30	44
59	1908 May 12	Luzerne	do.	1,100	850	-	do.	Feeders ignited and extinguished	do.	Gas carried out over men moving it	Supervision, no open lights in moving gas, and remove men when moving gas	12	11	-	31	45	57
60	1909 Mar. 2	do.	Slope	2,800	1,500	-	Local	Gas found in face, brattice men clearing it	do.	Used safety lamps, one had open light	Supervision, no open lights in moving gas, and remove men when moving gas	8	6	-	21	22	30
61	1910 Jan. 11	do.	do.	2,500	1,100	-	do.	Fall destroyed check or curtain left 16 hours	do.	Ventilation restored moved gas over open light	Remove men till section is clear, flame safety lamps	7	0	6	-	-	-
62	1910 Mar. 12	do.	Shaft	2,500	1,200	7	do.	Idle 3 days - repairs "shorted" air	do.	Fire boss reported clear; crew went in	Inspection, no open lights	7	0	0	17	20	34
63	1911 Jan. 10	do.	do.	700	350	-	do.	Gas from abandoned breast forced on gangway	do.	Built ventilation wall moved gas over open light	Ventilation, inspection, closed lights	3	1	-	-	-	-
64	1911 May 27	Northumberland	Slope	1,400	900	-	do.	Cut into gassy bed	do.	Men left but followed fire boss in with open lights	Supervision, closed lights	5	-	-	23	28	36
65	1912 Jan. 9	Luzerne	do.	700	400	-	Strong	Settling roof held door open	do.	Ignition set off boxes of dynamite and black powder	Closed lights, restrict amount of explosives in any one place	6	2	1	-	-	-
66	1912 July 17	do.	Shaft	2,500	1,200	-	Local	Poor ventilation of faces	Blasting	Black powder and dynamite fired electrically	Ventilation and permissible explosives	4	0	-	28	35	45
67	1913 Apr. 12	do.	do.	3,500	1,600	10	do.	Deranged ventilation	Open lights	Mixed lights	Ventilation, inspection, closed lights	4	3	-	-	-	-
68	1913 July 22	do.	do.	2,000	1,100	-	do.	Gas ignored; used locked safety lamps	Smoking	5 men in this known gassy section	Ventilation, no smoking	3	2	0	-	-	-
69	1913 Aug. 2	Schuylkill	do.	1,400	900	20	Strong	Pillar failure released gas and short-circuited air	Open lights	Men leaving met gas	Supervision, closed lights	20	0	0	25	28	55
70	1914 Sept. 16	Carbon	do.	1,500	900	-	do.	Booster fan stopped overnight	Smoking	Mixed lights	Eliminate booster fans; no smoking	7	2	-	29	40	47
71	1915 Feb. 17	Luzerne	do.	2,000	1,000	-	do.	Gas from high cavities	Open lights	Brought down by a "slide in"	Ventilation, closed lights	13	3	0	15	20	33
72	1916 Feb. 8	do.	Slope	1,500	700	-	Local	Open door	do.	Gas moved over lights when door was closed	Inspection, closed lights, continuous ventilation by use of overcasts, etc.	7	2	0	-	-	-
73	1916 Mar. 9	do.	do.	1,200	650	-	Strong	Large gas body released by blasting	Blasting, permissible and dynamite	Possible ignition by Davy lamp	Permissible shooting, approved lamps	6	1	-	-	-	-
74	1916 Aug. 8	do.	Shaft	3,500	1,600	10	Violent	Development work only, insufficient ventilation	Defective flame safety lamp	Working on new airways	Ventilation, inspection, closed lights	6	1	-	-	-	-

1/ Years marked with an asterisk are those in which there were no major accidents, or in which no accidents occurred.

Table 1.- Gas explosions in anthracite mines of Pennsylvania - Continued

Case	Date	County	Type of opening	Tons per day	Men employed	Men in mine	Extent of explosion	Cause of accumulation	Source of ignition	Other factors	Possible means of prevention at time	Killed	Injured	Escaped from area uninjured	Other cases: Ignitions	Other cases: Killed	Total yearly gas fatalities
75	1916 Aug. 9	Luzerne	1 slope 3 shafts	2,000	1,200	-	Small	Gas from borehole; poor ventilation	Open lights	Ignited by shift coming in	Ventilation, inspection, closed lights	2	3	-	-	-	-
76	Dec. 1	do.	Shaft	-	-	-	Strong	No ventilation, enlarging shaft	Blasting (50 per cent dynamite)	Reopening, all men on surface	Ventilation, inspection, permissible blasting	2	1	-	24	30	53
77	1917 Dec. 31	Lackawanna	3 shafts	1,300	600	-	Local	Open door	Open light	Man met gas which was being moved	Supervision, closed lights	1	9	-	31	44	45
	1918* 1/	-	-	-	-	-	-	-	-	-	-	-	-	-	25	28	28
78	1919 July 8	Carbon	Slope	3,000	1,600	-	do.	Gas released by blasting	Unknown	Smoking, or misuse of safety lamp	Ventilation, closed lights	8	3	-	26	40	48
	1920*	-	-	-	-	-	-	-	-	-	-	-	-	-	34	41	41
79	1921 Mar. 9	Schuylkill	Shaft	1,600	800	-	do.	Booster fan stopped overnight	Electric arc	Electricians moved fan and threw in switch	Test for gas, eliminate booster fans	5	6	3	-	-	-
80	Nov. 7	do.	Slope	1,300	650	-	Small	Closed regulator from idle section	Smoking	"Closed light" mine	Supervision	4	0	-	44	56	65
	1922*	-	-	-	-	-	-	-	-	-	-	-	-	-	10	15	15
81	1923 Feb. 21	do.	Shaft	1,100	600	-	Local	Brought down raise by blasting	do.	Poor ventilation and inspection	Ventilation, inspection, prohibit smoking	5	-	-	-	-	-
82	June 26	Northumberland	Slope	1,300	875	-	do.	Double doors too close to contain trip	do.	Left open for trips to pass	Proper spacing of doors, no smoking	5	-	-	30	35	45
83	1924 June 6	Luzerne	Shaft	3,000	1,250	-	do.	Replacing broken doors	do.	When broken doors replaced, gas moved over men	Inspection and supervision	14	6	-	36	42	56
84	1925 May 22	do.	do.	5,000	2,200	-	do.	Open door	Electric arc	Cable-reel locomotive	Ventilation, permissible equipment	7	1	-	-	-	-
85	June 8	Carbon	Slope	3,500	1,700	-	do.	Irregular ventilation	Blasting	"Sand blast" of 25 pounds Red H permissible	Permissible blasting and adequate ventilation	4	-	-	-	-	-
86	Aug. 3	Luzerne	do.	1,900	900	-	do.	Open door	Smoking	When door was closed gas moved to slope	Ventilation and supervision	10	2	-	18	22	43
87	1926 May 6	Schuylkill	do.	450	230	-	do.	Ventilation by compressed air only	Blasting (40 per cent dynamite)	By blasting or premature explosion	Ventilation and permissible blasting	5	4	-	-	-	-
88	June 29	Luzerne	Shaft	8,000	1,750	-	Violent	Door open while shifting cars	Defective flame safety lamp	"Motor" crew rerailing car	Eliminate doors, inspect flame safety lamps	3	8	-	-	-	-
89	July 3	do.	Slope	2,200	1,200	-	Strong	Pillar fall	do.	Gas forced over men working	Ventilate gobs, inspect flame safety lamps	7	8	-	-	-	-
90	Sept. 2	Northumberland	do.	1,200	600	-	do.	Blasting blocked intake	Smoking	Gangway on return air	Ventilation and supervision	4	-	-	-	-	-
91	Oct. 30	Luzerne	Shaft	3,500	2,200	-	do.	Fans stopped over holiday	do.	Starting booster fans pulled gas over men	Eliminate booster fans, no smoking	9	-	-	23	30	58
92	1927 Apr. 27	do.	do.	1,000	360	-	Local	Lack of brattices	do.	Gas gathered in hole in roof	Ventilation, no smoking	4	-	-	-	-	-
93	May 26	do.	do.	4,500	1,600	-	Violent	Squeeze and fall	Unknown	Gas ignited and started fire, causing second and third explosion	Proper supervision	7	20	-	-	-	-
94	Sept. 29	Lackawanna	do.	3,500	1,600	-	Local	Open doors	Electric arc	Open-type haulage locomotive	Doors in pairs, permissible equipment	4	5	-	-	-	-
95	Oct. 7	Schuylkill	Slope	1,200	600	-	do.	Released by blasting	Open light	Only safety and electric cap lamps allowed by rules	Proper supervision	4	1	-	33	45	64
96	1928 May 25	Luzerne	Shaft	2,000	900	-	do.	Ventilation interrupted	Smoking or electric arc	Gas moved over men at lunch	Ventilation, no smoking, permissible equipment	10	5	-	-	-	-
97	July 9	Schuylkill	Slope	1,500	750	-	do.	Face of incline not ventilated	Smoking	Closed-light mine; smoking or open light	Ventilation and supervision	3	0	4	17	19	32
	1929*	-	-	-	-	-	-	-	-	-	-	-	-	-	27	31	31
98	1930 Feb. 7	do.	do.	2,000	900	-	do.	do.	Open light	No inspection before men went up	Ventilation, inspection, closed lights	4	0	-	-	-	-
99	Feb. 25	do.	Shaft	1,500	800	-	do.	Released by blasting	Blasting	Overcharged shot, permissible explosive, 10-shot battery	Inspection, permissible shooting	4	0	-	-	-	-
100	Sept. 17	do.	Slope	600	190	-	do.	Feeder and poor ventilation	do.	Blow-out mixed shot, nonpermissible battery	do.	4	2	-	-	-	-
101	Sept. 29	Luzerne	Shaft	1,000	600	-	Strong	Falls released gas, short-circuited ventilation	Open lights	"Nongassy" mine, men ignited gas when going in	Ventilation, inspection, closed lights	4	3	0	-	-	-
102	Oct. 10	do.	do.	1,200	500	-	do.	Feeder and poor ventilation	Davy lamp	Left lighted in gas which burned through gauze	Ventilation and permissible lamps	4	1	-	16	22	42
103	1931 Jan. 14	do.	Slope	175	70	-	Local	Took boards from brattice for door	Smoking	Closed lights, gas moved over men	Instruction and supervision	3	-	-	-	-	-
104	May 29	Northumberland	do.	2,000	1,000	-	do.	A fall interrupted ventilation	Blasting	Blow-out shot, permissible explosive, 10-shot battery	Permissible shooting methods	5	-	-	-	-	-
105	Sept. 4	Luzerne	do.	3,000	1,000	-	do.	Feeders, poor ventilation	Smoking	Gas moved over men when line brattice was replaced	Ventilation and supervision	3	1	-	-	-	-
106	Oct. 24	do.	Drift	1,500	760	-	Strong	Tapped old workings	do.	6 men entombed, 2 died of burns, 2 of carbon monoxide	Supervision	4	2	-	7	8	23
107	1932 June 1	Schuylkill	Slope	-	-	-	Local	Booster fan, poor ventilation	Blasting		Proper ventilation	2	-	-	-	-	-
108	July 14	Luzerne		-	-	-	do.	Line brattice about 50 feet outby face, poor ventilation	Smoking	Failure of bosses to detect gas and presence of gas ignored by chargeman	Improved ventilation, better testing for gas	2	1	-	-	-	-
109	July 26	do.		-	-	-	do.		Open light		Closed lights	1	-	-	-	-	-
110	Aug. 13	Schuylkill	Shaft	16	7	-	do.	Poor ventilation, gas released by rush of coal, leakage from unventilated section	Smoking	Auxiliary fan	Proper ventilation, removal of auxiliary underground fans, permissible blasting devices	0	5	2	0	0	5
		Total		149,266	73,847	229						599	294	61	2/ 1,122	1,614	2,213

1/ Years marked with an asterisk are those in which no major accidents occurred, or in which no accidents occurred.

2/ Ignitions totaled 1,232: 110 "representative" ignitions and 1,122 other ignit

CAUSES OF EXPLOSIONS IN ANTHRACITE MINES

The probable causes of the 110 representative gas explosions are summarized in Table 3; the four principal causes of the explosions are open lights, smoking, explosives or blasting devices, and flame safety lamps. Causes are discussed under separate headings.

Table 3. - Causes of ignitions of gas in anthracite mines in 110 selected explosions

Cause	Explosions
Open lights	56
Smoking	18
Explosives or blasting devices	14
Flame safety lamps	12
Mine fires	4
Electric arc or spark	3
Furnace	1
Unknown	2
Total	110

Open lights.- Of the 110 cases listed, 56 were caused by open lights, and 25 of the 32 explosions occurring prior to 1890 were caused by open lights. Although the need of a safe lamp was recognized, the Davy lamp, which was invented in 1816, gave too feeble a light to make it popular. About 1890 safety lamps became more generally used and explosions from open lights became less frequent, although 9 such explosions occurred in the 10-year period from 1890 to 1899, inclusive. During the next 33 years only 22 explosions attributed to open lights occurred. One explosion was caused by a fire boss exploring unventilated workings while carrying an open light.

Safety lamps.- Davy lamps caused 12 of the listed ignitions. In 5 of these the lamps were opened or were in a defective condition; in the other 7 ignitions, flame came through the gauze as a result of leaving the lighted lamp too long in an explosive mixture or in too strong a current of air. The first record of an explosion from a Davy flame safety lamp in a Pennsylvania anthracite mine was in 1870 and the latest in 1930

Smoking.- Although smoking was practiced in earlier years, the presence of open lights eliminated it as a source of ignition because the open flame was already present. Unquestionably numerous gas ignitions were caused from smoking where flame safety lamps were used, before 1906, when the first ignition charged to smoking appears in the list; from 1906 electric cap lamps or permissible locked flame safety lamps were used extensively, so that smoking was marked as doubly dangerous and far more reprehensible, as the use of closed lights almost invariably outlaws or prohibits smoking. Of the 110 gas ignitions recorded in Table 1, 18 explosions were charged to smoking from 1906

to 1932, inclusive; it is significant that four of the eight explosions in the anthracite mines of Pennsylvania listed for 1931 and 1932 were attributed to smoking.

Fires.- In 4 cases, fires which had been started by open lights or blasting, ignited or reignited gas when explosive mixtures re-formed.

Explosives or blasting devices.- The first case of an ignition from blasting as given in Table 1 was in 1890; from that date until 1932 there are 14 cases listed. Blasting with nonpermissible blasting units caused some of these explosions and black blasting powder, dynamite, or mixed explosives were responsible for several explosions.

Electricity.- The first important ignition from an electric arc was in 1921, another is listed in 1925, and a third in 1927; this source of ignition had been suspected in several other cases where the evidence was inconclusive. In at least 5 explosions listed under "explosives or blasting devices" the most probable cause was nonpermissible electric blasting units that gave off an electric spark.

Unknown causes.- In two other explosions no evidence of the source of ignition was found, but in both an open flame is indicated. All of the 110 cases cited were thought to be ignitions and explosions of methane; it would appear that this hazard should be fully recognized, but in spite of this knowledge, open lights, smoking, nonpermissible blasting devices and explosives, and "safety" lamps that have proved themselves unsafe have been allowed to remain in some anthracite mines. Probably 100 of the 110 explosions listed in Table 3 were caused by open type or other recognized unsafe equipment.

CAUSES OF GAS ACCUMULATIONS IN ANTHRACITE MINES

The need for properly controlled ventilation as a prerequisite in preventing gas explosions is evident, but many explosions are the direct result of violating good ventilation practices. The causes directly accountable for the accumulation of gas responsible for the 110 explosions cited have been correlated in Table 4, showing the number of accumulations by major causes and subdivided into minor causes. The largest number of accumulations was the direct result of insufficient or interrupted ventilation. Falls or slides of roof or coal rank second in causes of accumulations of gas, open doors rank third, feeders and outbursts of gas fourth, blasting fifth, and fans sixth. The major groups that cause gas accumulation are discussed in more detail to emphasize the underlying causes of failure to remove or dilute explosive gas in the anthracite mines of Pennsylvania.

Table 4.- Causes of gas accumulations in Pennsylvania anthracite mines - 1847 to 1932

Causes of accumulations	Number
Insufficient or interrupted ventilation	47
Changes in ventilation	13
Insufficient ventilation	13
Ventilation not carried to face	8
Lack of ventilation	6
Ventilation interrupted	3
No brattice or brattice not in place	3
Gas from bore hole not diluted	1
Falls or slides of roof or coal	18
Squeeze or fall blocked airway	7
Fall released gas	5
Gas released by run of coal	4
Fall in old workings forced out gas	1
Squeeze and fall released gas	1
Open doors	14
Door left open	8
Removal of door	1
Settling roof held door open	1
Door in stopping left open	1
Double doors, too close to contain trip, left open	1
Repairing broken door	1
Door open while shifting cars	1
Feeders and outbursts of gas	11
Feeders at face	3
Gas in old workings released when tapped in mining	3
Gas burst from coal	2
Feeders in test hole	1
Feeders in sealed area	1
Gas released when gassy bed was cut into	1
Blasting	8
Gas released by blasting	7
Blasting blocked intake airway	1
Fans	6
Booster fan idle overnight	2
Fan stopped 20 minutes with hot bearing	1
Fan stopped 1 hour	1
Fans stopped over holiday	1
Booster fan not delivering enough air	1

Table 4.- Causes of gas accumulations in Pennsylvania anthracite mines - 1847 to 1932--Continued

Causes of accumulations	Number
Miscellaneous causes	4
Sump filled and man entered past danger board	1
Old workings entered, not ventilated, no examination	1
Presence of gas ignored	1
Unknown	1
Change of ventilation by unauthorized persons	2
Regulator closed	1
Boards removed from brattice to use in door	1
Total	110

Insufficient or interrupted ventilation.- Probably all failures to "dilute, render harmless, and remove gas" could be attributed to insufficient ventilation; however, the 47 cases segregated under this heading are due wholly to the causes named. Changes in ventilation, causing 13 accumulations, consisted of changing ventilation pipes, removing air hose during blasting, changing regulators, erecting brattice, extending line brattice, and similar methods to remove bodies of gas. A like number of accumulations, 13, were caused by failure to supply sufficient air to remove gas as it was liberated. Gas accumulated at faces of workings that were not ventilated prior to 8 explosions of gas. Failure to ventilate or seal abandoned or idle workings caused 6 accumulations of gas that resulted in explosions. Interruption of ventilation and failure to put up brattices or to replace brattices each caused 3 gas accumulations. One explosion was caused by failure to dilute gas issuing from a bore hole.

The United States Bureau of Mines considers all coal mines gassy because even where little or no gas has been detected, sudden outbursts or release of gas has occurred. Therefore, air should be so conducted that a strong current will sweep every working face and enough air will be conducted through all open workings to prevent gas accumulations. In removing gas accumulations, safe practice demands that all persons be removed from the mine or at least from the return-air side of the gas accumulations during the moving process.

Falls or slides of roof or coal.- Eighteen gas accumulations were caused by falls or slides of roof or coal which disrupted the ventilation or released ~~ing~~ quantities of methane. Squeezes or falls, blocking airways, resulted in gas accumulating prior to 7 explosions. Falls of roof or coal released large volumes of gas in 5 cases, run of coal released gas in 4 cases, a fall in old workings released gas in 1 case, and a squeeze and a fall of roof also released gas in 1 case. Where it is known or even suspected that gas may be released in case of falls, squeezes, coal runs, etc., good mining practice should demand that possible sources of ignition such as open lights, blasting, electric sparks or arcs should be held to a minimum.

Open doors.- In 14 cases, open doors resulted in gas accumulations; doors being left open caused 8 of these accumulations and an accumulation was caused in each case by removal of a door, roof settling and holding a door open, a door in a stopping being left open, doors in an air lock being too close to allow a trip to pass without both doors being open, and a door left open while cars were being shifted.

Feeders and outbursts of gas.- In some mines a large amount of the gas is liberated from feeders, and in 11 of the cases cited, feeders or outbursts of gas caused accumulations; at least 5 of these were the result of uncovering feeders, and the other 6 were caused by gas rushing from old workings, or from outbursts from gas in beds under pressure.

Blasting.- In many mines considerable gas is released by blasting and at times feeders or bodies of gas are released; in 7 of the cases cited release of explosive gas by blasting was the cause of accumulations, and in one case blasting blocked an airway, cutting off ventilation. By all means blasting should, if possible, be done when the working shift is not in the mine or at least not in the immediate vicinity or on the return side of the place or places where blasting is done.

Fans.- Failure, in some manner, of fans caused at least 6 accumulations of gas; 5 of these were due to shutting down of so-called auxiliary fans, and 1 was caused by a booster fan not delivering enough air. Auxiliary fans used underground are, in most cases, an admission that the ventilating system is inefficient.

Changes in ventilation by unauthorized persons.- Changes in ventilation by unauthorized persons is a far too common cause of inadequate mine ventilation, although Table 1 cites only two cases: tampering with a regulator and demolishing a brattice to obtain timber. It is possible that a number of the cases listed under insufficient ventilation may be attributed to this cause.

Miscellaneous causes.- In 3 of the 4 miscellaneous cases of gas accumulations the presence of gas was known, and precautions were taken to warn workmen, who disregarded the warnings. Disregarding danger signs is by no means an unusual occurrence; however, these violations clearly show the need of ventilating <u>all</u> places that are not sealed, and also the need of education of the workers and of more effective supervision by management.

PREVENTION OF EXPLOSIONS IN ANTHRACITE MINES

Explosions in Pennsylvania mines are generally investigated by State mine inspectors and by trained safety investigators of the United States Bureau of Mines. The recommendations made by these men are based on wide experience and mature judgment; therefore such suggestions should be given full consideration. The methods recommended for preventing the explosions in anthracite mines as given in Table 1 are listed in Table 5.

Table 5.- Methods recommended for preventing explosions listed in Table 1 in anthracite mines at the time of the explosions

Recommendations	Times recommended
Ventilate mine or improve ventilation	39
Use or proper use of flame safety lamps	39
Inspection, or improve inspection methods	29
Supervision	22
Use permissible (closed) lights	18
Use permissible explosives and blasting units	9
No smoking	8
Eliminate auxiliary fans underground	3
Instruction of employees	3
Use permissible electric equipment	3
Remove men from return air when moving gas	2
Eliminate doors by building overcasts	2
Use only safety lamps in moving gas	2
Proper spacing of doors in air locks	2
Precautions in fighting mine fires	1
Remove workmen from mine after erecting seals	1
Seal fires that cannot be fought directly	1
Provide second or stand-by fan on surface	1
Prohibit matches underground	1
Ventilate all open workings	1
Use double doors	1
Blast only when men are off return air	1
Make true reports of gas detected	1
Remove men until section is clear of gas	1
Restrict amount of explosives in one man's possession	1
Test for gas	1
Ventilate gobs	1

Ventilation.- Greatest emphasis has been placed on adequate ventilation, this recommendation having been made 39 times in the reports on the 110 explosions listed in Table 1 and probably should have been made in many additional instances. It has been shown that failure to ventilate, interruption of ventilation, failure to conduct air to the faces or failure to provide sufficient air allowed gas to accumulate in 78 of the 110 cases cited. In 16 others pockets or feeders of gas were released, by cutting into abandoned areas or gassy beds or by blasting, in amounts so large that the normal air currents could not properly dilute the gas; here the obvious remedy is that in mines where gas may be expected in fairly large quantities, ignition agencies should be eliminated or at least very definitely restricted or safeguarded. In 6 instances, standing gas in unventilated old workings was ignited in place or when moved over open lights.

In view of these facts the logical conclusion is that all open parts of the mine should be ventilated by a colume of air that will not only "dilute, render harmless, and remove" gas under normal conditions, but also will be sufficient to "render harmless" sudden outbursts of gas; this requires the elimination of doors wherever possible, construction of tight stoppings, erection of effective line brattices, and construction of tight overcasts. All abandoned areas should be securely sealed, if practicable, and if they cannot be sealed they certainly should be adequately ventilated.

Flame safety lamps.- The use of flame safety lamps in explosion prevention evidently has been considered of equal importance with ventilation, each recommendation being made 39 times (see Table 5). The Davy flame safety lamp has been by no means a dependable safeguard, but when carefully used it helped to reduce probability of occurrence of explosions in its day. In 1886, a State inspector cites a case of two men killed by a burst of gas and coal: "Due to use of safety lamps, there was no explosion, which saved the loss of all the men in the mine that night." With the development of the modern approved flame safety lamp and electric cap lamp, explosions due to ignition by open lights should have ceased, though they have not done so by any means.

The Davy flame safety lamp has served its purpose; now, safer and more effective lamps are available, and unquestionably they should be used. At least one explosion was caused by an employee disassembling a flame safety lamp; if the up-to-date or permissible flame safety lamp is used, its magnetic lock should prevent unauthorized opening. The up-to-date permissible flame safety lamp also has double gauzes wherever gauzes are used, and prevention of gas ignition is not dependent on a single gauze in some parts as is the case with most Davy lamps; the propagation of flame to the surrounding gassy atmosphere by a Davy lamp, as was the case in one of the explosions listed, probably was the result of using a single gauze. Flame safety lamps must be handled properly and should be used only by a carefully trained employee; if carefully manipulated they may serve as a means to detect explosive gas and prevent explosions; unless extreme care is exercised in their use, flame safety lamps become an additional explosion hazard. Hundreds of lives have been sacrified in coal-mine explosions in the United States, and in other countries as well, started through misuse of flame safety lamps.

Inspection.- Table 5 shows that in 29 cases inspection or improved inspection methods were recommended as a means of preventing explosions which have been listed in Table 1 as having occurred in the anthracite mines of Pennsylvania. Hundreds of ignitions, resulting in fatal injuries, have occurred in Pennsylvania anthracite mines when men, many of them officials and fire bosses, went into unventilated, uninspected workings with open lights.

Inspection to be effective must be a conscientious, inflexible, rigid, well-planned procedure, not to be disregarded or slighted by officials or by workers. Numerous explosions have been started by fire bosses or other officials who carried flame safety lamps, simultaneously wearing open lights, and entering uninspected, unventilated workings. To prevent gas ignitions

in anthracite mines, inspectors should be assured that their eyesight is sufficiently acute to determine a gas cap and then should test, thoroughly, with a properly assembled permissible flame safety lamp with a properly adjusted flame, and under no circumstances should open flame lamps be lighted or electrical equipment (especially open-type electrical equipment) be operated or shots be fired until the absence of dangerous, gassy mixtures is assured. The proper coursing of the air currents should be determined while testing for gas is in progress and careful inspection should be made as to sources of ignition, such as open flames, or electric arcs and sparks, to eliminate explosion hazards at their sources. Searching employees for matches, igniters, and smoking materials should be an important duty of inspectors and the search should be rigid, thorough and if possible made under surprise conditions, as many explosions are started by deliberate violation of "no smoking" rules.

Supervision.- In Table 5 supervision ranks fourth in the number of recommendations made for preventing explosions. Supervision is the basis of all safety efforts and is wholly responsible for effectiveness of ventilation, for the presence or absence of sources of ignition, and for inspection to determine presence of matches, igniters, open lights, the use of nonpermissible explosives and blasting devices, and nonpermissible electrical equipment or of permissible or nonpermissible equipment in a state of poor repair.

Study of data in the foregoing tables concerning explosions which have occurred in anthracite mines reveals that the number of fatalities from explosions varies with the number of ignitions in the ratio of approximately 2 to 1; in this, incipient ignitions or explosions in anthracite mines are by no means as destructive as in bituminous mines.

A notable decrease in fatalities from explosions followed supervision of anthracite mines by State mine inspectors from 1870 to 1888; an increase in fatalities then followed, essentially paralleling increase in production from 1889 to 1927, broken only by extended idle periods from strikes in 1902, 1906, 1922, and 1925. The introduction of permissible (closed) lights in the more dangerous sections of the mines of some of the more progressive companies may be responsible in part for the temporary lowering of the number killed in explosions from 1916 to 1923. The marked decline in the fatality rate from explosions in anthracite mines since 1927 (see Table 2) has been due largely to improved ventilation and to the extension of the use of permissible electric cap lamps, of permissible flame safety lamps, of permissible explosives, and of electric firing of blasts; also, it has probably been aided by more intensive supervision in an effort to conform more closely to the State law, by the added influence of compensation measures, and by the pressure of economic conditions.

Permissible (closed) lights.- Approximately half of the explosions listed in Table 1 were caused by open lights, yet the use of closed lights was recommended only 18 times; undoubtedly failure to recommend closed lights prior to 1910 was due to the fact that in those days about the only closed light available was the flame safety lamp with its poor illuminating features and clumsiness for ordinary use. It is interesting to observe that electric (safety)

cap lamps were first used and developed in the United States in the anthracite mines by the Philadelphia & Reading Coal & Iron Co. during 1908. This company, according to J. T. Jennings, "awoke to the dangers and disadvantages of the open-flame miners' lamp, and considering the benefits in safety, efficiency, and cleanliness an electric lamp would have over the ordinary open-flame oil lamp in the hands of a miner" made a "canvass of the United States (that) showed that no electric miners' lamp existed at that time." The electrical department of the company then developed a satisfactory lamp. The permissible electric cap lamp, if used in conjunction with adequate ventilation should eliminate from anthracite mines any gas explosions caused by open lights.

In 1912, one large company had 1,500 safety lamps in use, after 5 years of pioneer experimentation; in 1932, approximately 33,000 electric cap lamps were in use in anthracite mines.

Smoking.- At least 18 explosions have been caused by smoking, yet only 8 recommendations to prohibit smoking and the carrying of smoker's accessories such as matches and patented lighters or igniters have been made in reports on the 110 explosions listed in Table 1; it is significant that two of the explosions from smoking occurred in 1932. As a source of ignition in anthracite mines, it is as important as open lights and other open-type equipment; therefore, smoking should be prohibited in all anthracite mines--in fact, in any underground mine; and the most determined effort should be made to enforce the rule at all times, particularly as regards smoking by mine officials.

Permissible explosives and blasting units.- As long as open lights were used, few ignitions resulted from blasting, because the lights generally ignited the gas before the shots were fired, although gas released by blasting was occasionally ignited by the shots, and in numerous instances gas liberated by shots and carried by air currents was ignited by open lights. With growth of the use of closed lights, ignitions through or at time of blasting became more common. The introduction of permissible explosives and electric blasting helped to limit these occurrences in anthracite mines, but the use of fuse, squib, and nonapproved blasting units, and overcharged and tight holes or holes with little or no stemming left loopholes, and explosions due to blasting continue to occur. Electric blasting is safer than firing shots with fuse or squib, but explosions have been caused by nonpermissible multiple blasting units or by using the power lines in blasting. The growth of permissible explosives in anthracite mines is shown in the following tabulation.

Types of explosives used in anthracite mines, pounds

Year	Black blasting powder	Dynamite	Permissible explosives
1917	35,754,225	16,074,125	7,298,925
1931	10,389,300	15,569,820	15,608,235

EXPLOSIONS IN BITUMINOUS MINES

Practically from the time of the earliest settlement in Pennsylvania, coal has been mined from outcrops for local and domestic use; the garrison of Fort Pitt mined coal on the banks of the Monongahela River in 1760 for its own use, and in 1786 mining was done under the authority of William Penn in the same locality. In 1845, some coke was produced and used in making iron. In 1842, coal production was about 500,000 tons, in 1850 about 1,000,000 tons, and in 1877, when the first bituminous inspectors were appointed, about 14,000,000 tons.

Mines were mainly drifts from the outcrops, with but little gas hazard, though in 1877 and 1878 several ignitions were reported which did not result in fatalities.

Table 6 lists 170 explosions of various kinds in the bituminous coal mines of Pennsylvania of which record is available from 1878 to 1932, inclusive; of these, 85 are described in more or less detail as being explosions in which there was marked violence, or in which several were killed by burns or afterdamp. Small ignitions are grouped in the last three columns of Table 6 as other cases in the yearly totals.

Fatalities[5]

Available records as listed in Table 6 extending from 1878 to Jan. 1, 1933, show that 170 gas and dust explosions in Pennsylvania bituminous mines resulted in 1,984 fatalities or an average of about 12 fatalities in each explosion; the number killed in any one explosion ranged from 0 to 239. In scattered cases the injuries are considerable, but compared with the deaths, the number is small except possibly in the earliest years.

Table 7 presents the gas and dust fatality rates per ton and per million man-hours of surface and underground work; the rates are remarkable for the wide variation and in particular for the fine showing in the last two years. The rate based on man-hours follows the fatality rate in general trend, though it reflects the influence of increased efficiency in production. To 1895, the ratio of the two rates is about 3 to 1, then is approximately 2 to 1 until 1925; from that date the ratio gradually decreases, dropping to about 1.5 to 1 in 1930 and the ratio is 2 to 1 for the entire period 1879 to 1931, inclusive. Gas and dust explosion fatalities averaged 12 per cent of the total killed in the bituminous mines of Pennsylvania over the entire period 1879 to 1931, inclusive.

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5 See reference 4, page 2.

Table 6.- Gas and dust explosions in bituminous mines of Pennsylvania, 1878-1932

Case	Date	County	Type of mine	Tons per day	Men: Average	Men: In mine	Explosion: Type	Explosion: Extent	Explosion: Limited by	Cause of accumulation	Source of ignition	Other factors	Possible means of prevention at time	Killed	Injured	Escaped from area uninjured	Other cases: Ignitions	Other cases: Killed	Total yearly fatalities
	1878* 1/	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0	0
1	1879 Dec. 29	Westmoreland	Drift, gassy	90	150	Few	Gas	Small	Amount of gas	Leaky stoppings, open door	Open light	Men going in, no inspection	Ventilation, inspection, flame safety lamps	3	-	-	2	2	5
	1880*	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1	1	1
2	1881 Oct. 17	Washington	do.	225	135	-	do.	Local	do.	Feeder and open door	do.	Door closed, gas forced over lights	Supervision and flame safety lamps	1	-	2	2	2	3
	1882*	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	5	8	8
	1883*	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1	1	1
3	1884 Feb. 20	Fayette	Shaft, new	300	90	-	Gas (?)	Moderate	Little dust, weak start	Poor ventilation and inspection	do.	Gas ignored in idle rooms	Ventilation, inspection, flame safety lamps	19	1	-	-	-	-
4	Apr. 14	Allegheny	Drift, gassy	100	175	2	do.	do.	do.	Open door, no inspection	do.	Men going to work	Inspection and flame safety lamps	2	0	0	-	-	-
5	Oct. 27	Fayette	Slope, gassy	300	105	-	do.	Strong	Probably some dust fired	Open door, no inspection, idle section	do.	Door closed, gas forced over lamps	Ventilation and inspection of idle workings, flame safety lamps	14	4	2	0	0	35
	1885*	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1	1	1
6	1886 Mar. 8	do.	do.	150	65	25	do.	do.	Little violence	Standing gas in idle mine	do.	Cut through from this mine	Care in approaching old workings, flame safety lamps	5	10	-	-	-	6
	1887*	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	3	5	5
7	1888 Nov. 3	Clinton	Drift, new, no gas	150	130	-	Dust	do.	Low violence and expansion	Post fell on box of detonators, set off dynamite and black powder	Dynamite explosion	First recognized dust explosion in Pennsylvania	Keep detonators and explosives apart	17	2	5	1	1	18
8	1889 May 10	Allegheny	Drift, gassy	500	200	4	Gas and dust	Violent	Lack of oxygen	Mine idle, fan stopped 10 days	Open light	4 men went to get tools	Supervision and flame safety lamps	4	0	0	2	2	6
	1890*	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0	0	0
9	1891 Jan. 27	Westmoreland	Shaft, gassy (1885)	1,000	450	-	do.	Small	Low violence and impure dust	Fall and feeder after fire boss inspection	do.	Shift going in; most killed by carbon monoxide	More inspection and flame safety lamps	109	-	-	2	2	111
	1892*	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1	1	1
	1893*	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1	1	1
	1894*	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0	0	0
	1895*	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	4	4	4
10	1896 Mar. 23	Clearfield	Shaft, gassy (1893)	100	60	13	do.	do.	do.	Feeder ignited by shot, blew itself out	Fire	Explosive mixture set off by coals	Inspection, care of fires	13	0	0	-	-	-
11	June 29	Fayette	Slope, gassy	700	170	2	do.	Violent	Impure dust	Mine idle, fan stopped 20 days	Open light	2 men went in for tools by 2 danger boards	Supervision and flame safety lamps (?)	2	0	0	2	2	17
	1897*	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1	2	2
12	1898 Sept. 23	do.	Drift, gassy	500	130	-	Gas	Light	Small volume	Fan run part time only	do.	Gas over falls reported but not removed	Removal of gas and use of flame safety lamps	8	-	-	3	4	12
13	1899 July 24	do.	Shaft, gassy	800	115	-	do.	Small	do.	Idle room, gas reported but not removed	do.	Mixed lights, man went over danger board	do.	5	2	-	-	-	-
14	Dec. 23	do.	Shaft, new, no gas	210	80	30	Gas and dust	Violent	Wrecked shaft and one side of mine	Fan stopped or door open	do.	No inspection before men entered	Inspection and flame safety lamps	19	4	6	3	5	29
15	1900 Nov. 20	Washington	Shaft, gassy	80	75	-	do.	Strong	Weak start, expansion	Poor ventilation of faces	Blasting	Hole on solid fired in gas against orders, dynamite	Ventilation and safer shooting practice	3	2	-	3	5	8
16	1901 Mar. 25	Fayette	do.	350	200	-	Gas	Local	Small volume	Careless ventilation	Blown-out shot	Shot in gas and against orders, dynamite	do.	4	0	-	-	-	-
17	June 10	Westmoreland	do.	500	110	-	Gas and dust	2 explosions and fire	Low violence and expansion	Ventilation short-circuited by squeeze, gas ignited, blew out stoppings	Open light	First killed 4 men, set fire, second killed 15 rescuers	Ventilation and flame safety lamps	19	0	0	3	5	28
18	1902 Mar. 6	Washington	Drift, gassy	1,000	225	-	Gas	Strong	Lack of oxygen	Behind fire seals	Mine fire	Opened seals in 48 hours	Leave fires sealed until extinguished	5	2	-	-	-	-
19	July 10	Cambria	do.	1,900	550	-	Gas and dust	Small	Low violence, expansion	Unventilated old workings	Open light	Moved into working area by pillar removal	Ventilation or sealing of old workings, flame safety lamps	112	-	18	-	-	-
20	Dec. 24	Fayette	do.	1,000	170	4	Gas	Strong	On entry, lack of dust	Mine and fan stopped 5 days, danger board up	do.	Men went in without inspection	Ventilate and inspect before men go in, flame safety lamps	4	0	0	4	5	126
21	1903 May 26	Allegheny	Drift, new, no gas	350	125	-	do.	Small	Small volume expansion	Poor face ventilation and door off 2 weeks	do.	No inspection	Ventilation, inspection, flame safety lamps	4	3	-	-	-	-
22	Nov. 21	Fayette	Drift, gassy	.375	110	20	do.	Light	do.	Sealed fire area	Sealed fire	Pillar removal let air in, killed by CO	Leave sufficient pillars	17	3	0	1	2	23
23	1904 Jan. 25	Allegheny	Shaft, gassy	450	280	180	Gas and dust	Violent	2 men outside and 2 rescuers killed	Ice choked air shaft, poor inspection	Blown-out shot	No open lights, dusty, dynamite shot electrically	Inspection, dust control, safer shooting	179	1	0	3	5	184
24	1905 Apr. 27	Clearfield	do.	2,100	800	-	do.	do.	Lack of dust and expansion	Open door	do.	Black powder, no inspection	Safer shooting	13	1	0	-	-	-
25	July 6	Fayette	Slope and shaft, new, no gas	20	58	-	Gas	Strong	In shaft	Platform for concreting 72 feet up	Open light	No ventilation or inspection	Ventilation and closed lights	6	-	-	-	-	-
26	Oct. 29	Washington	do. gassy	600	200	-	do.	Moderate	Low violence	Sealed fire area	Sealed fire	Opened 2 seals	Precautions with fires	5	1	-	-	-	-
27	Nov. 15	do.	do. no gas	0	15	10	do.	do.	In shaft	Platform put in 50 feet down	Open light	No ventilation or inspection	Ventilation, inspection, closed lights	7	3	-	4	6	37

1/ Years marked with an asterisk are those in which there were no major accidents, or in which no accidents occurred.

Table 6.- Gas and dust explosions in bituminous mines of Pennsylvania, 1878-1932 - Continued

Case	Date	County	Type of mine	Tons per day	Men: Average	Men: In mine	Explosion: Type	Explosion: Extent	Explosion: Limited by	Cause of accumulation	Source of ignition	Other factors	Possible means of prevention at time	Killed	Injured	Escaped from area uninjured	Other cases: Ignitions	Other cases: Killed	Total yearly fatalities
28	1906 Oct. 24	Cambria	Drift, gassy	3,000	850	-	Gas	Small	Small volume	Care left in door	Blown-out shot	Dynamite shot with electric battery	Split ventilation and safe explosives	7	0	0	4	4	11
29	1907 Dec. 1	Fayette	Shaft and slope, gassy	1,500	340	34	Gas and dust	Violent	Expansion	Poor ventilation and inspection	Blown-out shot	Black powder, possibly arc from trolley	Permissible explosives, ventilation, inspection	34	0	0	-	-	-
30	Dec. 19	Westmoreland	do.	1,500	400	-	do.	do.	do.	do.	Open light	Went into fenced-off gassy room	Ventilation and closed lights	239	1	0	4	4	277
31	1908 Apr. 23	Washington	Shaft, gassy	1,500	600	-	Gas	Local	Small volume	Open door	Open lights	Men coming out, pumper went in	Split ventilation and closed lights	4	-	0	-	-	-
32	June 19	do.	do.	1,500	600	-	do.	do.	do.	Released by fall	do.	Moved over pillar workers	Inspection and closed lights	3	9	-	-	-	-
33	Nov. 28	do.	Shaft, mixed lights, gassy	350	225	155	Gas and dust	Violent	Throughout mine	Poor face ventilation	Blown-out shot	Nonpermissible explosives and coal-dust stemming	Permissible shooting, ventilation and inspection	154	1	0	1	1	162
34	1909 Jan. 25	Somerset	Slope, gassy	700	320	-	Gas	Local	Low violence	Feeders when pillar work started	Open light	Ignited on day shift, night shift going in	Inspection and closed lights	5	-	-	-	-	-
35	Jan. 29	Westmoreland	do.	700	300	-	Gas and dust	Small	Low force and dampness	Small amount from floor crack	do.	Ignited by machineman	Testing by cutters, closed lights	2	2	1	-	-	-
36	Apr. 9	Somerset	Drift, no gas	350	150	-	Dust	do.	Expansion and impure dust at entries	Shooting top for overcast	Blown-out shot	Dynamite overcharged and misplaced	Wet dust, safer shooting	7	0	0	-	-	-
37	June 23	Indiana	Shaft, slope, no gas	1,300	500	-	do.	Strong	Wet sections and expansion	Hole blown out night before	do.	Shot 2 sticks of dynamite untamped	Safer shooting	21	12	-	-	-	-
38	Oct. 31	Cambria	do.	1,500	600	28	do.	do.	do.	Idle day, miner shot 2 holes off solid	do.	Shot dynamite in 1 foot of coal	do.	16	1	-	2	2	53
39	1910 Feb. 5	Indiana	Drift, gassy	700	200	-	Gas and dust	do.	Wetness	Idle room, gas allowed to stand	Open light	Miner went in	Inspection, ventilation, and closed lights	11	8	-	-	-	-
40	Oct. 16	Allegheny	do.	350	100	0	do.	do.	Wetness and low force	Fan shut down after shooting, for repairs	Probably burning feeder	Blew up 15 hours later	Inspection and spare fan	0	0	0	3	4	15
41	1911 July 15	Jefferson	Shaft, open light, gassy	1,000	270	29	do.	Violent, one side of mine	Wetness and impure dust	Unventilated face	Blasting	Mixed safety explosives and black powder	Ventilation, inspection, and safer shooting	21	0	0	-	-	-
42	Nov. 9	do.	Slope, gassy	1,500	600	-	do.	Strong	Little or no available expansion	Poor ventilation	Electric arc	Trolley wire under fall	Ventilation and wet dust	8	0	4	0	0	29
	1912* 1/		-	-	-	-	-	-	-	-	-	-	-	-	-	-	1	1	1
43	1913 Apr. 23	Washington	Drift, gassy	700	200	167	do.	Violent	Use of water and salt and expansion	Brushing gas out of room to fire shot	Open light	Apparatus man killed by CO	Closed lights and proper ventilation	98	0	3	0	0	98
44	1914 Oct. 26	Allegheny	Slope, open lights, gassy	1,500	415	0	do.	Strong	Expansion and patches of water	Fan idle over Sunday, power thrown on	Electric arc	Pump motor on, recovery worker killed	Continuous ventilation, rock-dusting	1	0	0	1	2	3
45	1915 May 24	Cambria	Slope, no gas	350	100	8	do.	do.	Weak start	Rock shot or poor ventilation and door open	Open light	Apparatus man killed by CO	Ventilation, closed lights and rock-dusting	9	0	0	-	-	-
46	July 1	Fayette	Slope, gassy	400	100	-	do.	600 ft., small	Weak start, very wet mine	Idle section poorly ventilated	do.	Pumper went in, no inspection made	Ventilation, inspection, closed lights, and rock-dusting	1	0	0	-	-	-
47	Aug. 31	Somerset	do.	1,500	500	-	Gas	do.	Small amount	Cavity in roof	Arc from trolley	Gas distributed when door opened	Ventilation, restrict power wires	19	8	-	1	1	30
48	1916 Feb. 11	Indiana	Drift, mixed lights, gassy	2,000	350	-	Gas and dust	do.	Weak start and dampness	Released by pillar falls	Open light	Falls not inspected	Closed lights, inspection, rock-dusting	27	4	-	-	-	-
49	Mar. 30	Westmoreland	Shaft, slope, new, no gas	150	38	8	do.	Entire mine	Entire mine	Poor ventilation and no inspection	do.	Possibly another source	Ventilation, inspection, closed lights, rock-dusting	8	0	0	-	-	-
50	Oct. 21	Jefferson	Slope, gassy	2,200	400	0	Gas and dust	Strong	Expansion and wet areas	Idle, one fan stopped, insufficient ventilation	Electric arc	Usually 100 men in part affected	Cut off current when idle, and rock-dust	0	0	0	1	1	36
51	1917 Mar. 13	Washington	Shaft, gassy	950	225	-	do.	do.	Impure dust and dampness	Open door and poor inspection	do.	Nipping, traveling machine	Split ventilation, inspection, storage-battery machine, rock-dust	14	-	24	-	-	-
52	Mar. 18	Fayette	do.	1,500	400	5	do.	do.	Dampness and expansion	Raised curtain and hole in pillar	do.	Nipping or pump motor	Proper stoppings, inspection, and rock-dust	4	0	0	-	-	-
53	Apr. 3	Westmoreland	Slope, non-gassy	100	41	3	do.	do.	Dampness on mains and expansion	Lack of air at faces, no inspection made	do.	Nonpermissible mining machine	Test by cutters, permissible machines, rock-dust	3	0	0	0	0	21
54	1918 Mar. 16	Armstrong	Drift, non-gassy	20	7	7	Dust	do.	Entire mine except air courses	Shot off solid, coal-dust tamping	Blown-out shot	Black powder and squib	Permissible shooting and rock-dust	4	0	3	-	-	-
55	Aug. 7	Allegheny	Shaft, small, non-gassy	50	30	-	Gas	do.	Wet, just reopened and pumped out	Almost no ventilation	Open light	No inspection	Ventilation, inspection, closed lights	8	0	0	0	0	12

1/ Years marked with an asterisk are those in which there were no major accidents, or in which no accidents occurred.

Table 6.- Gas and dust explosions in bituminous mines of Pennsylvania, 1878-1932 - Continued

Case	Date	County	Type of mine	Tons per day	Men: Average	Men: In mine	Explosion: Type	Explosion: Extent	Explosion: Limited by	Cause of accumulation	Source of ignition	Other factors	Possible means of prevention at time	Killed	Injured	Escaped from area uninjured	Other cases: Ignitions	Other cases: Killed	Total yearly fatalities
56	1919 Jan. 20	Clearfield	Slope, nongassy	25	20	-	Gas	Small	Small volume of water	Water rose and cut off air shaft	Open light	Moved into entry by blasting	Inspection and closed lights	3	-	-	-	-	-
57	1919 Mar. 17	Westmoreland	Drift, gassy	1,500	250	-	do.	Local	Floor covered with water	Door open	do.	Lack of inspection	Inspection, split ventilation, closed lights	3	-	-	-	-	-
58	1919 Oct. 8	do.	Drift, nongassy	800	220	-	do.	do.	Low violence	Pillar cave broke surface gas line	do.	Gas pulled in by exhaust fan	Mapping and care of pipe lines	2	2	12	1	2	10
59	1920 June 2	Washington	Sinking shaft, gassy	0	6	6	do.	Violent	Shaft just reached coal	Closed lights and poor ventilation	Unknown	Probably smoking	Ventilation, inspection, supervision	6	0	0	-	-	-
60	1920 June 7	Indiana	Drift, gassy	125	50	-	do.	Local	Mine very wet, reopening	Stoppings broken, repairing	Open light	Closing holes moved gas over crew	Closed lights and supervision	2	3	-	-	-	-
61	1920 June 13	Armstrong	Drift, nongassy	350	125	2	do.	do.	Small volume	Fan stopped Saturday, run Sunday with open door	do.	Machine crew, no inspection	Inspection and closed lights	2	0	0	-	-	-
62	1920 July 19	Allegheny	Sinking shaft, gassy	1,700	325	9	Gas and dust	Violent, entire mine	Entire mine	Fan repairs, 2 days	Electric arc	Fire boss took "motor" in on run	Inspect before men enter and rock-dust	9	0	0	3	3	22
63	1921 May 23	Cambria	Drift, nongassy	600	100	-	do.	Strong	Dampness and expansion	Door open and lack of inspection	Open light	Men going in	Split ventilation, inspection, closed lights, and rock-dust	2	-	-	0	0	2
64	1922 Feb. 2	Fayette	Shaft, gassy	2,500	400	29	do.	do.	do.	Idle day, gas in face, no testing	Electric arc	Firing wires, third shot gas and dust	Permissible firing, ventilation, rock-dust	25	0	0	-	-	-
65	1922 Mar. 20	Indiana	Drift, gassy	700	150	-	Gas	Small	Low violence	Fan idle Sunday, fire boss skipped section	do.	Closed lights, open-type room hoist	Inspection and permissible equipment	5	3	-	-	-	-
66	1922 Nov. 6	Cambria	Shaft, nongassy	500	117	-	Gas and dust	Moderate	Low violence and wet patches	Poor ventilation and inspection	Open light	Moved over lights (?)	Closed lights, ventilation, inspection, and rock-dust	77	22	11	1	1	108
	1923* 1/	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0	0	0
67	1924 Jan. 26	Indiana	Slope, gassy	200	50	47	do.	Strong	Expansion and inert dust on old roads	Faces ahead of air, poor ventilation and inspection	Mining machine (electric arc)	Resistance of nonpermissible machine	Ventilation, testing, permissible machines, rock-dust	36	-	-	-	-	-
68	1924 Mar. 17	Armstrong	Drift, nongassy	15	5	3	Dust	Small	Expansion and lack of dust	Shooting off solid	Blown-out shot	Black powder	Permissible explosives and rock-dust	3	0	0	-	-	-
69	1924 July 25	Fayette	Shaft, gassy	2,300	350	-	Gas and dust	Strong	Active sprinkling	Opened abandoned section unventilated	Electric arc	Fall forced gas out to cable nip	Ventilation, scaling roof, compressed air or storage battery equipment, rock-dust	10	-	-	2	2	51
70	1925 Apr. 26	Westmoreland	Slope, sinking, gassy	0	7	5	Gas	do.	Shaft only	Gas ignored, used flame safety lamps and electric bulbs	do.	Light bulb or wires	Ventilation, permissible electrical equipment	5	0	0	1	1	6
71	1926 Feb. 3	Allegheny	Shaft, gassy	3,700	600	400	Gas and dust	do.	Rock-dust and expansion	Sealing fire from feeder and machine	do.	Open-type machine, no testing	Testing, ventilation, permissible machine	20	0	2	-	-	-
72	1926 June 1	Westmoreland	Drift, gassy	175	55	1	do.	Moderate	Low violence and expansion	Fan stopped off and on, door open	Open light	Fire boss alone in mine	No open lights, proper ventilation	1	0	0	-	-	-
73	1926 Aug. 26	Indiana	Slope, gassy	300	70	52	do.	do.	do.	Interrupted ventilation	Electric arc	Gas moved over broken trolley or blower-fan motor	Split ventilation and rock-dust	44	4	-	3	4	69
74	1927 Mar. 30	Cambria	3 drifts, gassy	1,700	350	294	Dust	6000 ft., strong	Inert dust and expansion	Coupling broke and let 36 cars loose	do.	Power cable, 275 volt, wheel hit splice	Remove cables from haulway, rock-dust	4	-	17	-	-	-
75	1927 Apr. 2	Washington	Shaft, gassy	1,800	350	306	do.	Strong	Rock-dusted entries only	Green man clearing roof fall	Blasting	2 mud-capped shots of dynamite	Permissible blasting, thorough rock-dusting	6	5	2	0	0	10
76	1928 Feb. 20	Westmoreland	Shaft, slope, gassy	1,800	500	-	Gas and dust	5000 ft., strong	Dampness and expansion	Fall or open door	Electric arc	Nipping mining machines in return air	Split ventilation, no wires in return, rock-dust	12	-	7	-	-	-
77	1928 Apr. 11	do.	Shaft, gassy	1,100	125	-	do.	Strong	Rock-dust, water, and expansion	Interrupted ventilation	do.	Machine cable	Permissible equipment, water on cutting bar, rock-dust	4	1	-	-	-	-
78	1928 May 19	Greene	do.	3,800	600	270	do.	9000 ft., violent	Little rock-dust, expansion	Open doors	do.	Nonpermissible storage-battery locomotive	Ventilation, permissible equipment, rock-dust	195	-	14	-	-	-
79	1928 Aug. 9	Cambria	Slope, gassy	70	14	10	do.	Small	Low violence and dampness	Machine cut into feeder	do.	Nonpermissible mining machines	Permissible equipment, water, rock-dust	5	0	5	-	-	-
80	1928 Aug. 15	Clearfield	Drift, gassy	550	145	-	do.	Strong	Small volume of water and expansion	Lack of inspection and ventilation	do.	do.	Inspection, ventilation, permissible equipment, rock-dust	13	1	-	1	1	230
81	1929 Mar. 21	Westmoreland	Shaft, slope, gassy	2,000	350	263	Dust	Violent	Partly rock-dusted	Steel conveyor broke in slope (intake)	do.	Cut power cable	Cables out of haulways and rock-dust	46	4	-	1	1	47
82	1930 May 15	Washington	3 shafts, 2 slopes, gassy	1,900	450	-	Gas	Local	Low violence and expansion	Feeder released by pillar fall	do.	Motor or trolley, fire, abandoned	Ventilation, supervision, permissible equipment	3	4	-	-	-	-
83	1930 Aug. 5	Armstrong	Drift, nongassy	350	-	48	do.	do.	Small volume	Poor ventilation	Open light	Lack of inspection	Ventilation, inspection, permissible lights	1	2	-	1	1	5
84	1931 Aug. 28	Clarion	do.	-	2	1	Oil vapor	do.	Volume	Broken oil line near surface	do.	Unexpected occurrence	Provide maps showing pipe-lines	1	0	0	0	0	1
85	1932 July 29	Somerset	Shaft, mixed lights	2,500	360	60	Gas and dust	do.	Volume and expansion	300 feet ahead of air	Probably conveyor motor (electric arc)	Ventilation, inspection	Ventilation, inspection	3	1	1	0	0	3
	Totals			75,780	19,980	2,540								1,876	137	139	2/ 85	108	1,954

1/ Years marked with an asterisk are those in which there were no major accidents, or in which no accidents occurred.
2/ Ignitions totaled 170: 85 "representative" ignitions and 85 other ignitions.

Table 7. - Gas and dust explosion fatalities in Pennsylvania bituminous mines, 1879-1931

Period, years	Fatalities per million ~~tons~~		Percent of explosion fatalities to total killed in Pennsylvania bituminous mines
	Tons	Man-hours	
1879 - 1880	0.17	0.05	5.8
1881 - 1885	.38	.12	12.3
1886 - 1890	.21	.06	7.0
1891 - 1895	.52	.17	15.5
1896 - 1900	.22	.09	6.5
1901 - 1905	.80	.32	18.4
1906 - 1910	.76	.32	18.0
1911 - 1915	.20	.09	6.6
1916 - 1920	.12	.06	4.6
1921 - 1925	.25	.13	9.3
1926 -	.45	.25	16.5
1927	.08	.04	2.8
1928	1.75	.99	44.6
1929	.33	.19	12.3
1930	.04	.03	1.7
1931	.00	.00	0.0
Total 1879-1931	0.335	0.169	12.0

CAUSES OF EXPLOSIONS IN BITUMINOUS MINES

The causes of explosions in bituminous mines are restricted essentially to four: open lights, resulting in 40 explosions; electric arcs, or sparks, resulting in 24 explosions; explosives or blasting devices, initiating 16 explosions; and mine fires, contributing to 4 explosions. Three causes that were listed under explosions in anthracite mines in Pennsylvania do not appear in Table 8. No explosion is known to have been initiated by flame safety lamps in the bituminous mines of Pennsylvania, although 12 explosions were caused by Davy flame safety lamps in anthracite mines. No explosions were attributed to smoking in bituminous mines, although smoking caused 18 explosions in anthracite mines. No explosions in bituminous mines are known to have been caused by furnaces.

Table 8. - Causes of explosions in Pennsylvania bituminous coal mines, 1878 - 1932

Causes	Number of explosions
Open lights	40
Electric arc or spark	24
Explosives or blasting devices	16
Mine fires	4
Unknown	1
Total	85

Open lights.- Of the 85 explosions listed in Table 6, 40, or 47 percent were caused by open lights. During the 4 years July 1, 1927, to June 30, 1931, open lights caused at least 32.5 percent of all coal mine explosions in the United States; therefore, in Pennsylvania bituminous mines the percentage of explosions by open lights was 14.5 greater than in this 4-year average in the coal mines of the United States, but during this 4-year period, open lights caused only 14.2 percent of the explosions in Pennsylvania mines.

Open lights caused a large percentage of the explosions until 1920, but in the 13 years from 1920 to 1932 there were only 7 explosions from this source. The widespread use of permissible electric cap lamps in Pennsylvania bituminous mines after 1920 undoubtedly is largely responsible for the decrease of explosions from this source.

Electricity.- From 1911 to 1932, inclusive, 24, or 28 percent, of the explosions are charged to electrical ignitions. Since 1924, of 17 explosions listed, 13 or about 77 percent are of this origin. In 1894, a State inspector's annual report included a warning of the increasing hazard from the introduction of electrical equipment into gassy mines. At least in part as a result of special inspection of electrical equipment by the State in 1929 to see that it was maintained in accordance with the standards established by the United States Bureau of Mines, such explosions were reduced to one small ignition in 1930, none in 1931, and one in 1932. This is a real achievement, one of which Pennsylvania bituminous mining men may well be proud and one which could well be studied and followed by coal mining people in other States. The following table gives the types of explosions in Pennsylvania.

Table 9. - Types of explosions in bituminous mines

Condition	Gas	Dust	Gas and dust	Total
Nongassy mines . . .	9	6	7	22
Gassy mines.	24	4	35	63
Explosions	~~23~~ 33	10	42	85
Fatalities	189	132	1,555	1,876

The classification of mines as gassy or nongassy was rather vague until about 1909; from that year until 1924, the nongassy mines figure prominently in the list of mines in which explosions occurred. However, there have been numerous other ignitions resulting in fatal and nonfatal injuries in this class of mine. The United States Bureau of Mines considers all coal mines gassy or potentially gassy and has records of scores of cases in which so-called nongassy mines have almost overnight given off gas in large quantities; in numbers of such cases severe explosions have resulted.

The 85 explosions listed in Table 8 caused 1,876 fatalities; the other 85 unclassified explosions resulted in 108 fatalities. Dust was the largest factor in the fatalities, as the explosions in Pennsylvania bituminous mines in which gas only was involved averaged about 6 killed, even including the most severe gas explosions, while the average number killed where dust or gas and dust were involved was 35.

Blasting and explosives.- One explosion of dynamite in 1888 caused a dust explosion, which was probably the first recognized dust explosion in this country. Blown-out shots and other dangerous blasting risks caused 14 other explosions from 1900 to 1928, most of them prior to 1910 and practically all of them involving dynamite or black blasting powder.

Fires.- Of the 4 ignitions from fires, 1 was from smoldering coals and killed 13 and 3 were from sealed fires opened too soon, costing 27 lives. Mine fires are always a potential cuase of explosions, and when fires occur they should be given the most careful handling.

CAUSES OF GAS ACCUMULATIONS AND DUST SUSPENSIONS IN BITUMINOUS MINES

In 76 explosions gas ignitions were the primary cause and in 9 explosions gas played no appreciable part insofar as can be ascertained, coal-dust alone being the igniting and propagating medium. The largest number of gas accumulations came from insufficient or interrupted ventilation. Open doors ranked second in the cause of gas accumulations, fans and falls of roof or coal were third, and feeders or inrush of gas fourth. Dust was raised into a cloud and ignited by shooting off the solid and other methods of blasting, by cars or conveyors wrecking on slopes, and by a post falling on explosives that in turn detonated. These primary causes of explosions are discussed under the major divisions and are summarized in Table 9.

Table 9. - Causes of gas accumulations and dust suspensions in Pennsylvania bituminous mines

Causes	Explosions
Insufficient or interrupted ventilation	33
Insufficient ventilation at face	11
Insufficient ventilation, other than at face	7
No ventilation	6
Interrupted ventilation	2
Gas found but not removed	2
Careless ventilating practice	1
Gas in cavity at face	1
Raised curtain, hole in pillar	1
Water flooded section cut off air in shaft	1
Repairing stopping	1
Open doors	16
Door left open	14
Door removed	1
Cars left in doorway	1
Fans	10
Fan idle; moved gas when started	7
Fan not operating	3
Falls of roof or coal	10
Fall of roof released gas	4
Pillar removal and falls	2
Fall broke gas or oil pipe line on surface	2
Squeeze caused short-circuit of air	1
Fall of roof or open door	1
Feeders and inrush of gas	6
Feeders opened by pillar removal	2
Feeders released by cutting coal	2
Gas rushed in from abandoned mine	1
Feeder from floor	1
Miscellaneous	3
Removing fire seals before fire extinguished	3
Gas released by blasting	1
Dust raised into suspension	9
Shooting off solid	3
Blasting	3
Trip of cars wrecked on slope	1
Broken conveyor on slope	1
Post fell on explosives	2

Insufficient or interrupted ventilation.- Of the 85 explosions listed in Table 6, 33 were caused by ignition of gas that accumulated through insufficient or interrupted ventilation. Insufficient ventilation at the face resulted in 11 accumulations, and 7 accumulations resulted from insufficient ventilation in places other than the face. "No ventilation" allowed gas to accumulate prior to 6 explosions. Interrupted ventilation and gas found but not removed were each responsible for 2 gas accumulations. Other causes were the result of careless ventilating practices, gas in cavity in roof not removed, a curtain raised or a hole in a pillar short-circuiting the air, water cutting off air in a shaft, and failure to provide temporary brattice while repairing a stopping.

Open doors.- In 14 explosions, gas that was ignited had previously accumulated through doors being left open. The removal of a door and cars left in a door each were responsible for gas accumulations.

Fans.- Fans should be operated at all times, and failure to do so may result in gas accumulations; saving money by shutting down the fan on idle days is likely to prove expensive economy in the long run. Shutting down fans in 7 instances allowed gas to accumulate; explosions followed, in each instance when the gas was moved over a source of ignition when the fans were started. In 3 cases gas accumulated while fans were not operating.

Feeders or inrush of gas.- Sudden inrushes of gas occur when feeders are released when coal is cut, or when pillars are removed, or when falls occur. Falls or removal of barrier pillars may release gas in an adjoining abandoned mine and force it into the working property. Feeders were released and ignited in 6 cases, as recorded in Table 6; the best protection against occurrences of this type is the elimination of sources of ignition such as open lights, nonpermissible electrical equipment, etc.

Other causes.- In 3 explosions, gas that had accumulated when fires were sealed was diluted to form explosive mixtures and was ignited by smoldering fires when seals were removed before the fires were extinguished. Blasting released gas and ignited it in at least one instance.

Dust raised into suspension.- Nine of the explosions were classed wholly as dust explosions and 7 of these were caused by blasting or explosives. In 3 explosions dust was raised into a cloud and ignited by explosives where coal was shot off the solid. Blasting caused combustible dust to be raised into the air and ignited by the explosives in 3 instances, 1 when blasting rock for an overcast, 1 when coal was blasted without stemming being placed in the shot hole, and 1 when mud-capped shots were fired.

A trip of cars wrecked on a slope threw dust into suspension, and an arc from the trolley wire ignited the dust. The breaking of a coal conveyor threw into suspension a dense cloud of coal-dust and which was ignited by an electric arc.

PREVENTION OF EXPLOSIONS IN PENNSYLVANIA BITUMINOUS MINES

The recommendations for preventing a recurrence of similar explosions were made in reports of the various explosions. These represent, in general, the accepted safety practices at the time they were made, and as such are of interest. The recommendations and the number of times each was made are given in Table 10.

Table 10. - Recommendations for preventing explosions at time of explosions in bituminous mines of Pennsylvania

Recommendations	Times made
Ventilation	44
Inspection	33
Dust control (watering, rock-dusting, water on cutter bar)	31
Permissible (closed) lights	22
Flame safety lamps	15
Permissible explosives and blasting devices	15
Permissible or storage-battery equipment	12
Supervision	6
Testing for gas by machinemen	3
Inspection of idle workings	2
Care with fires	2
Removal of gas	2
Maps of and care with surface gas and oil lines	2
Remove cables from haulage road	2
Care in approaching old workings	1
Keep detonators and explosives apart	1
Leave fires sealed until they are extinguished	1
Seal old workings	1
Leave sufficient pillars around fire area	1
Auxiliary fan	1
Restrict extension of wiring	1
Cut off power in idle sections or idle mine	1
Proper stoppings	1
Scaling roof	1
No electric wires in return airways	1

Ventilation.- The need for well-designed systems of ventilation is of even greater importance in bituminous mines than in anthracite mines, for an ignition of gas that may cause little damage and no injuries in anthracite mines may develop into a widespread explosion in bituminous mines through propagation by explosive coal-dust. This hazard and the need for taking precautions is reflected in the 44 times that recommendation for improving ventilation was made.

Fans adequate for the needs should be provided and the air should then be taken to the working faces with a minimum of loss. Some of the best-ventilated Pennsylvania mines conduct at least 90 per cent of the air to the working areas. This is accomplished by constructing tight, fireproof stoppings, providing large area openings for conducting air--generally accomplished by utilizing a number of entries for airways--by eliminating doors wherever possible through construction of overcasts, and by sinking auxiliary shafts or extending entries to the outcrops as the workings advance to provide fresh air near the active workings and simultaneously to reduce the amount of travel of the air with attendant decrease in power consumption by the fan.

Inspection.- The second largest number of recommendations, 33, was for improving methods of inspection, generally interpreted to mean that testing for gas should be more thorough and that closer attention should be given to the ventilating system to prevent gas from accumulating. While these are two of the important duties of an inspector or supervising official, it is equally important that he should maintain a constant survey of dust-control methods such as watering and rock-dusting, keep a close check on electrical equipment to see that the possibility of producing an external spark or arc is reduced to a minimum, and see that open-flame lamps are not used, smoking is not allowed, and that open-type equipment, if impracticable to prohibit underground, is restricted to use in strictly fresh air.

Dust control.- The control of dust by rock-dusting and watering was recommended 31 times, indicating that the hazard of dust in bituminous mines was recognized by the trained investigators, but that the mine operators disregarded or were unaware of the extreme hazard of combustible dust. Explosions may be controlled if rock-dust is applied in amounts large enough to render the combustible dust inert to explosibility and if all open parts of the mine are rock-dusted to within at least 50 feet of every active working place and to the face of every idle or abandoned, unsealed place. Watering should be practiced to allay the dust at the source; water should be applied on the cutter bar of mining machines during cutting, and the face region should be wetted before blasting, the coal pile should be kept wet during loading, and the tops of loaded cars should be wetted before the cars are hauled from the working place. Water should also be sprayed on loaded trips before leaving a section, and considerable success has been attained by sprinkling haulage-road beds as an adjunct to rock-dusting.

Rock-dusting has been advocated by various engineers both in and out of the Bureau of Mines in the United States from 1908 onward. About 1913 it was generally proposed by the Bureau of Mines engineers as a preventive measure, and in 1923 it was made an official recommendation of the Bureau of Mines, based on long series of tests in its Experimental mine, though it was used but little in commercial mines until after that date. In the last few years there have been numerous cases where serious disasters have been averted in mines so protected, and there must have been many more which have gone unrecorded and perhaps unnoticed. (See Bureau of Mines Inf. Circ. 6596, April, 1932.)

Permissible (closed) lights.- The fourth largest number of recommendations (22) was that permissible (closed) lights be used for miners' portable lighting. The danger of open lights has generally been recognized and most of the mine workers in Pennsylvania bituminous regions are provided with permissible electric cap lamps; nevertheless, large numbers of open lights are now being used, and in any mine whether gassy or so-called nongassy they will always be a source of ignition of dust, explosives, and possibly of gas.

The installation of closed lights, however, in many cases results in neglect of ventilation. This should never be allowed, as such neglect is very likely to result in explosions of gas ignited by electric arcs or smoking.

Flame safety lamps.- Under "Inspection" it was pointed out that testing for dangerous mixtures of gas and air is essential to prevent the accumulation of gas unknown to workmen in the mine. At present the flame safety lamp is one of the quickest instruments for determining the presence of gas and at the same time indicating oxygen deficiency. It is doubtful if the average person making tests for gas detects less than 3 per cent of methane, although by "capping" the flame, careful inspectors who have very good eyesight may detect less than 1 percent. The flame safety lamp is safe only when properly assembled and in the hands of a competent, carefully trained person. All flame safety lamps should be tested in a scientifically designed testing box; reliance should not be placed on time-honored but inefficient methods of testing flame safety lamps by blowing on or at them. All lamps should be permissible, magnetically-locked types, and those locks should not be tampered with.

Safety lamps and closed lights.- Electric cap lamps in use in the bituminous mines of Pennsylvania have increased from a number estimated to be 800 in 1913 to 106,640 in 1932. The rapid adoption of closed lights was undoubtedly responsible for the marked decrease in open-light explosions after 1920. The United States Bureau of Mines started testing electric lamps in 1909, and granted the first approval in 1913. The State inspectors were constantly striving for better ventilation and inspection since first appointed in 1877. They obtained cooperation of the operators to an extent that was responsible for rapid and continued improvements. The new code, put into effect in 1911, called for locked flame safety lamps approved by the United States Bureau of Mines, and permissible explosives when ordered by the inspectors.

Permissible explosives and blasting devices.- The hazards of black powder and dynamite are generally well known, and the effectiveness and safety of permissible explosives are generally recognized; nevertheless, nonpermissible explosives are still widely used and even where permissible explosives are used, black powder and long-flame dynamite are also used. The hazard of using mixed explosives is often greater than the use of nonpermissible explosives alone.

Electrical blasting is by far safer than is blasting by fuse or squibs though electrical blasting itself is by no means foolproof. Electrical

blasting devices should be of a permissible type, for tests have shown repeatedly that multiple and nonpermissible blasting devices will ignite gas. Unquestionably one means of preventing explosions in both dusty and gassy bituminous coal mines is to use permissible explosives in a permissible manner, and to blast with permissible blasting units.

Permissible explosives.- The types and quantity of explosives used in Pennsylvania bituminous mines follow:

Classes of explosives used in bituminous mines

Year	Black powder	Dynamite	Permissible
1901	7,900,000	1/ 676,000	- - -
1917	19,071,275	2,332,339	8,218,750
1931	[illegible],335,525	965,045	8,067,007

1/ According to State Mine Inspector's report.

The Rolling Mill explosion in 1902 fixed attention on the need of safer types of explosives; some early types were developed at this time and were widely used after the Harwick disaster in 1904. Through establishment of the United States ~~Bureau of Mines~~ Geological Survey testing station in 1908 at Pittsburgh, new and improved brands of explosives were placed on the market. Modern blasting devices are not foolproof, but by observance of known and proved limitations in their use, ignitions from this source can be prevented. However, there are several pertinent points still to be studied in firing practice and the development of blasting units, but unquestionably the best method of preventing widespread explosions with heavy loss of life through blasting in bituminous coal mines is to do all blasting work when the working shift is out of the mine.

Permissible storage-battery or other permissible equipment.- Twelve recommendations were made for the use of permissible equipment, including storage-battery equipment. The use of electrical equipment underground introduces numerous sources of gas and dust ignition. During the four years ended June 30, 1931, Bureau of Mines Information Circular 6540 shows that 45.8 percent of the mine explosions in the United States were of electrical origin and that the fatalities resulting from the explosions of electrical origin constituted 74.8 percent of all explosion fatalities. In other words, if electrical equipment had been permissible and maintained in a permissible condition, nearly three fourths of the explosion fatalities might have been prevented, as there is not on record an explosion known to have been initiated by permissible electrical equipment.

For mine use, permissible mining machines, rock-dusting machines, pump motors, telephones, junction boxes, switches, locomotives, lights and practically every type of electrical equipment needed in mines are available. Their use and proper maintenance will help to prevent explosions. The record

shows that nearly 77 percent of the explosions in the bituminous mines of Pennsylvania since 1924 have been initiated by electric arcs from nonpermissible electrical installations, and it is to be hoped that the vigilance of the past 3 years as to electrical equipment in Pennsylvania bituminous mines will not be relaxed.

Supervision.- The eighth recommendation, made only 6 times, is for adequate supervision. Unquestionably this recommendation could well have been made in numerous other cases. It is through supervision that ventilation is effectively maintained, inspection is kept efficient, dust is controlled, and adequate safeguards as to igniting agencies are provided; it is the backbone of explosion prevention, just as it is the keystone of the provention of accidents of all kinds and of efficient operation as well.

Miscellaneous recommendations.- Seventeen other recommendations are made, but each is a corollary to the main recommendations.

Fatalities and mine explosions.- A study of production and of data on the number killed in gas and dust explosions in bituminous mines of Pennsylvania from 1873 to 1932 indicates that there is essentially no correlation between production and fatalities or explosions. Naturally, where mines have been closed for considerable periods by strikes, the number of explosions drops. The small number of explosions in relation to the high cost in lives is clearly shown. The high peaks from great disasters are regularly followed by low points, showing that renewed precautions and preventive measures induced by such lessons are effective, until forgetfulness and slackness leaves another opening, or in some cases a new and unforeseen combination of agencies arises.

Tablo 11. - Number of explosions and fatalities from explosions in Pennsylvania mines, 1847 - 1932

Years	Anthracite mines		Bituminous mines	
	Number of explosions	Fatalities	Number of explosions	Fatalities
- 1869	1/ 4	1/ 13		
1870 - 1879	119	269	1/ 3	1/ 5
1880 - 1889	109	223	23	84
1890 - 1899	191	409	23	177
1900 - 1909	221	355	53	909
1910 - 1919	261	424	28	255
1920 - 1929	291	450	35	545
1930 - 1932	2/ 26 36	2/ 70	2/ 5	2/ 9
Totals	1232	2213	170	1984

1/ Two years only recorded.
2/ Three-year period.

Some essential facts on explosions and the fatalities from explosions in Pennsylvania coal mines have been arranged by decades from 1870 to 1932 in

Table 11. In the anthracite mines from 1890 to 1929, inclusive, the number of explosions increased at the rate of about 3 per cent; however, in the 3-year period 1930 to 1932, inclusive, there were only 36 explosions; at the rate of 120 in 10 years, this is a notable decrease from the 291 which occurred in the decade 1920 to 1929, inclusive. The fatalities ranged from 355 to 450 each decade from 1890 to 1929, or 35 to 45 per year; in the years 1930 to 1932, inclusive, the fatalities have been at the rate of about 23 per year, a definite improvement on the past but by no means an irreducible minimum of fatalities from explosions in anthracite mines.

In the bituminous mines from 1880 to 1929 there have been 23 to 53 explosions per 10 years, with fatalities ranging from 84 to 909. During the 3 years 1930 to 1932, inclusive, in 5 explosions 9 men were killed, an admirable record and one difficult to improve. The average number of explosions per year in the last 3-year period was at a rate about half that of the other periods, and the number killed was from one third to one tenth of the average for preceding periods.

The United States Bureau of Mines, through its Mine Safety Board, issues "decisions" on safety methods and practices. Such decisions as relate to explosion prevention methods are as follows:

Decision 1:

The Bureau of Mines recommends:

(a) In all coal mines the portable lamps for illumination be permissible, portable, electric mine lamps; and also

(b) In places where fire damp or black damp is liable to be encountered, a permissible magnetically-locked flame safety lamp for gas detection, or equivalent permissible device, be supplied to at least one experienced employee in each such place; and

(c) Any employee before being supplied with a permissible flame safety lamp be examined by a competent official of the mine to assure the man's ability to detect gas; and

(d) All coal mines whether classed as nongassy or gassy in any part, be supplied with magnetically locked, permissible, flame safety lamps, properly maintained and in sufficient number for all inspection purposes.

Decision 2:

In the interests of safety the Bureau of Mines recommends that for blasting in coal mines, permissible explosives, fired electrically, be exclusively used; and that as an aid to blasting, all coal which is feasible to cut, should be cut or sheared.

Decision 4:

In the interest of safety, the Bureau of Mines, Department of Commerce, recommends that auxiliary fans or blowers should not be used in coal mines as a substitute for methods of regular and continuous coursing of the air to every face of the mine.

Decision 5:

To prevent the propagation of mine explosion, the Bureau of Mines, Department of Commerce, recommends rock-dusting all coal mines, except anthracite mines, in every part, whether in damp or dry condition. It also recommends that rock-dust barriers be used to sectionalize the mine as additional defense; but these should not be regarded as a substitute for generalized rock-dusting.

Decision 6:

In the interest of safety, the Bureau of Mines, Department of Commerce, recommends that in coal mines all entries, rooms, panels, or sections that cannot be kept well ventilated throughout or cannot be inspected regularly and thoroughly, or that are not being used for coursing the air, travel, haulage, or the extraction of coal, be sealed by strong fireproof stoppings.

Decision 7:

In the interest of safety, the Bureau of Mines, Department of Commerce, recommends:

1. That the main intake and main return air currents in mines be in separate shafts, slopes, or drifts.

2. That the main intake shaft lining be of fireproof construction, and there be a minimum amount of inflammable material in or adjacent to the shaft.

Decision 9:

The Bureau of Mines, Department of Commerce, recommends in coal-mine ventilation practice the following specifications as to unit quantity and quality of air:

1. The quantity in cubic feet of pure intake air flowing per minute in any ventilating split shall be at least equal to 100 times the number of men in that split.

2. The quantity of air entering each unsealed place shall be at least 200 cubic feet per minute and as much more as may be necessary to properly dilute, and carry away inflammable or harmful gases which may be present.

3. The air shall be made to circulate continuously to the face in every unsealed place into which an appreciable amount of methane enters.

4. The air in any unsealed place shall be considered unfit for men if it shall be found to contain less than 19 percent oxygen (dry basis), more than 1 percent carbon dioxide, or a harmful amount of poisonous gas.

5. If the air in any unsealed place, when sampled or tested in any part of that place not nearer than 4 feet from the face and 10 inches from the roof, shall be found to contain:

(a) more than 1-1/2 percent of inflammable gas, the place shall be considered to be in hazardous condition and require improved ventilation, and

(b) if more than 2-1/2 percent of inflammable gas is found, the place shall be considered dangerous, and only men who have been officially designated to improve the ventilation and are properly protected shall remain in or enter said place.

6. If the air in the split which ventilates any group of workings contains more than 1-1/2 percent of inflammable gas, these workings shall be considered to be in a dangerous condition and only men who have been officially designated to improve the ventilation and are properly protected, shall remain in or enter said workings.

Decision 11:

In the interest of safety, the Bureau of Mines, Department of Commerce, recommends that in coal mines, haulage and (or) hoisting be kept in intake air as far as possible.

Decision 12:

The Bureau of Mines, Department of Commerce, extending Mine Safety Decision No. 2, recommends that for blasting either coal or rock in coal mines, permissible explosives or equivalent permissible device be used exclusively, and in addition recommends that in blasting

1. Each charge shall be in a hole properly drilled and stemmed with incombustible material.

2. Each shot shall be fired separately by a permissible single-shot blasting unit, using an electric detonator or igniting equivalent of a kind specified by the bureau for the particular permissible explosive or permissible blasting device.

3. Before and following each shot in gassy and slightly gassy coal mines, examination for gas shall be made with a permissible flame safety lamp or permissible equivalent and

4. If more than 1-1/2 percent of inflammable gas is found, in the quantity and by the method specified in Mine Safety Decision No. 9, the place shall be considered to be in a hazardous condition and before another shot is fired the gas shall be reduced by ventilation below the percentage and quantity specified in Decision No. 9.

5. Each shot employing explosives shall be prepared and fired by or under the immediate supervision of a man having a state certificate as a mine examiner, fire boss, or foreman, and whenever conditions permit all other men than those authorized to prepare and fire shots shall be out of the mine when shot firing with explosives is being done.

Decision 13:

The Bureau of Mines, Department of Commerce, recommends that when electricity is used in coal mines rated as gassy, or wherever in any mine the atmosphere may become gassy:

1. Electrical equipment shall be permissible.

2. Nonpermissible electrical equipment shall be used only in pure intake air.

3. Electrical power shall be cut off whenever the air in the workings is in a dangerous condition, due to inflammable gas.

Decision 15:

In the interest of safety in coal mining, the Bureau of Mines, Department of Commerce, recommends that, to lessen the coal-dust explosion hazard:

(a) Machine coal cuttings be wet as the cutting is being done.

(b) The coal face, and the working place 40 feet therefrom, shall be kept free of coal-dust by the use of water.

(c) The top of loaded cars in the working place shall be wet.

Decision 16:

In the interest of safety in coal mining, the Bureau of Mines, Department of Commerce, recommends that:

(a) Machine cuttings be removed from the cut.

(b) If the machine cuttings are of a character which would contribute to a dust explosion, they shall be sent out of the mine.

Decision 17:

To lessen the formation and distribution of coal dust in haulage ways, the Bureau of Mines, Department of Commerce, recommends that in bituminous and lignite coal mines:

(1) The mine cars should be constructed and maintained dust-tight.

(2) The coal should be so loaded that it will not shake off in haulage.

(3) The cars and loads should be so sprayed as to prevent dust being distributed along the haulage ways.

Decision 18:

In the interest of safety in coal mining, the United States Bureau of Mines recommends that:

(1) The foreman regularly in charge of underground operations and also any person who, in the absence of the foreman, may be placed in temporary charge should each have a certificate of competency from the State to act as mine foreman.

(2) The superintendent or person in responsible charge of the mine, to whom the mine foreman reports, should have a certificate of competency from the State which should be issued upon a showing of underground experience for a period of time as long as that required for a foreman's certificate and upon passing an examination including all technical questions asked in the examination required of foremen.

(3) These certificates should expire after some stated period of time, such as 5 years, and should be removed only after the applicant has again passed the examination required by the State.

Decision 20:

In the interest of safety in underground mining, the United States Bureau of Mines recommends that while driving tunnels or drifts and sinking or raising shafts or slopes, and also in their operation, there should be an adequate ventilating current wherever men work or travel.

CONCLUSIONS

The very good record made in reducing the number and the accident severity of explosions during the last 3 years in Pennsylvania coal mines is largely due to greater effort on the part of State mine inspectors and the closer supervision given by the inspectors. Many mines are idle, and hence the inspectors are able to visit the operating mines more frequently and thus maintain a closer supervision. Credit for the reduction in explosions also is due the mine operators and mine workers who, in spite of unfavorable economic conditions, have in general given closer attention to safety, and to educational work looking to the forwarding of safety.

The principal primary causes of explosions have been insufficient or interrupted ventilation, open ventilating doors, release of gas by squeezes, falls or runs of roof or coal, the use or rather misuse of auxiliary fans underground, intermittent operation of fans, use of nonpermissible explosives and blasting units, release of feeders or bodies of gas by blasting or cutting into gassy beds or abandoned workings, and, in bituminous mines, dust thrown into suspension by blasting or by wrecked trips and conveyors. The chief causes of ignition of gas or dust were open lights, explosives or blasting devices, electric arcs or sparks, smoking, defective or carelessly handled flame safety lamps, and mine fires.

Explosion prevention depends primarily upon effective supervision, adequate ventilation, the exclusive use of permissible electric cap lamps for portable lights, the use of magnetically locked, flame safety lamps for testing for gas and for light, blasting with permissible explosives and permissible blasting units, and the use of permissible-type electric equipment. These fundamentals become effective when the underground employees are instructed in them, and such instruction is best accomplished by maintaining an active safety organization.

I.C. 6711
May, 1933.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING LAWS OF NEW CALEDONIA AND FRENCH OCEANIA[1]

By Paul M. Tyler[2]

PREFATORY NOTE

This paper presents one of a series of digests of foreign mining legislation and court decisions which is being prepared in advance of a general report relative to the rights of American citizens to explore for minerals and to own and operate mines in various foreign countries. These interpretations of the laws of New Caledonia and French Oceania have been prepared from the best information available in Washington, but are released subject to correction and amplification, if necessary, by the proper American diplomatic and consular officers to whom they are being referred through the courtesy of the Department of State.

NEW CALEDONIA

INTRODUCTION

New Caledonia became a French coly about 1853. A penal settlement (400 square miles) was established at Nou Island in 1864, but since 1896 no convicts have been sent thither and the convict element in the population is rapidly decreasing. The island is 248 miles long and averages [illegible] miles in breadth. Dependencies of the Colony include a considerable number of nearby islands, comprising the Loyalty group and the Wallis Archipelago, the Isle of Pines, Futuna and Alofi, and the barren Huon Islands, 170 miles to the northwest of New Caledonia. There is a local Governor who is assisted by a Privy Council consisting of the Secretary-General, the Procureur-General (head of the Judicial Administration), the Superior Commandant of the Troops, the head of the Department of Domains and Colonization, and two notables of the Colony appointed by the President of France. There is also an elective Council General.[3]

1/ The Bureau of Mines will welcome reprinting of this article, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular ."
2/ Chief engineer, rare metals and nonmetals division, U. S. Bureau of Mines.
3/ Data mainly from the Statesman's Yearbook.

The island is mountainous and transportation is difficult. The geologic map shows a fairly large area of sedimentary formations, certain of which are reported to contain coal. The production of coal increased in recent years until it amounted to 21,000 tons in 1929. From the mining standpoint, however, the serpentine series occupying about one-third of the island are of chief importance. At various times New Caledonia has been the world's leading producer of nickel, cobalt, and chromite, and a variety of other minerals have been produced commercially--in part by convict labor.

SOURCES OF LAW

Due to the importance of its mineral production New Caledonia was equipped with well-developed mining legislation fairly early in its history as a French colony. Since 1913, however, when a new decree was issued, the French colonial policy has undergone a sensible evolution, particularly with respect to the granting of permanent concessions and oil and gas legislation. These changes, together with certain additional modifications, were embodied in a still later decree dated August 28, 1927. The following abstract is based upon this law and all references except as otherwise noted are to articles therein.

CLASSIFICATION

Mineral deposits are legally divided into two classes, mines and quarries. (Art. 2.) Quarries comprise deposits that may be worked for building materials, ballast, and fertilizers and analogous materials for improving the soil except nitrates and associated salts and phosphates. Peat bogs are classed as quarries. The title to minerals of this class is not separated from the surface rights, and quarries consequently do not come within the province of the mining law except with respect to inspection and other measures administered by the Mines Service with a view to preserving the security of the surface and the safety of workers. (Art. 3.)

All minerals not classed as quarries are classed as mines (art. 4) and are specifically divided into 5 categories as follows (art. 6):

1. Solid mineral fuels (combustibles); i.e., coal.
2. Rock salt, nitrates, and other salines and phosphates.
3. Nickel, chromite, cobalt, manganese, and iron.
4. Bitumen and liquid or gaseous hydrocarbons; i.e., oil and gas.
5. All other substances.

Questions of legal classification of a substance or of a deposit are decided by the Governor with the advice of the Chief of the Mines Service and the Consulting Committee on Mines. (Art. 8.)

OWNERSHIP AND PROPERTY RIGHTS

Quarries belong to the owner of the land. Right to exploit "mines" is acquired from the State.

A prospecting permit constitutes real property. It may be transferred but may not be divided or mortgaged.

A concession of a mine constitutes real property wholly distinct from ownership of the land. Except for certain limitations in the mining decree it is freely transferable in all respects like ordinary real estate. (Art. 9.)

A concession relating to a deposit of one substance covers all other substances of the same category, but a concession or prospecting license relating to substances of a different category may be instituted in favor of other persons for the same area. (Art. 6.)

Quarry products (such as stone) produced in the regular course of mining or prospecting may be utilized for mining purposes. (Art. 7.)

PERSONAL LICENSES AND RIGHTS OF FOREIGNERS

Article 10 provides that individuals or companies holding prospecting permits or concessions must satisfy the conditions contained in the French decrees of January 8, 1916, and February 27, 1924. The latter prohibits owning or exercising prospecting or mining rights without first obtaining a personal license, which license may be cancelled at the discretion of the Governor. The war-time decree of 1916 (art. 2 thereof) provides that companies formed for prospecting and mining must conform with the French laws and have their main office either in France or in French colonies.

Any holder of title to a prospecting permit or concession (or assignee) must be represented in the Colony by a resident agent who should be French or at least able to obtain a personal license. (Art. 13.)

Any transfer or modification of the title must be reported and supporting documents submitted to the Chief of the Mines Service. In the case of sale or foreclosure of an active prospecting permit the declaration must be signed by both parties to the agreement. No transfer is valid until receipt of the declaration has been acknowledged by the Chief of the Mines Service. (Art. 11.)

PROSPECTING PERMITS

General.- Exclusive prospecting permits are issued by the Chief of the Mines Service. Priority of registration of application confers prior right to a permit, subject to the rights previously granted within the same area and the payment of the requisite fees and taxes. (Arts. 19 and 21.) As indicated later, special permits are required for minerals of the fourth category (oil and gas).

A permit covering a certain mineral conveys similar rights to other minerals in the same category but holders of a permit to prospect for one class of minerals may not oppose an application for a permit covering minerals of another category. (Art. 20.) Opposition may be entered at any time, however, on the basis of prior claim.

Area.- The claim is in the form of a square with sides at least 300 meters but not over 5 kilometers in length, and oriented true north and south and east and west. (Art. 19.)

If the rectangle overlaps an area previously granted for prospecting or mining substances of the same category, the area under the new permit shall be reduced by the extent of such overlap. (Art. 20.) The applicant is obliged to accept such additions to his claim as may be granted to him at the time. (Art. 21.)

Application.- The application is deposited at the office of the Mines Service in Noumea; it may not be sent through the mails. (Art. 23.) The Chief of the Mines Service is given one month in which to examine the application and acknowledge its receipt. (Art. 21.)

The application must set forth (1) the full name, title, nationality, and usual domicile of the applicant, together with the name of his representative in the Colony, or, if a company, the name, main office, and name and address of its resident agent; (2) the chosen address at Noumea (see art. 13); (3) the number of the personal license (as required by the decree of February 27, 1924); (4) the category of mineral sought; (5) the length of the sides of the square claim applied for.

Separate application must be made for each area and for each category of mineral sought.

The application should be accompanied by a map or rough sketch (croquis) of the surface on a scale of 1:10,000, oriented to the true north and indicating the positions of the corners of the square with reference to permanent landmarks easily recognizable in the field or upon a designated published map. (Art. 23.)

An application for a prospecting permit may be rejected (and fee therefor returned) (1) because of serious errors that can not be corrected, (2) if it is considered as without object (covering lands already covered by existing grants), (3) for nonpayment of fees. (Art. 25.) Right of appeal to the Governor is granted (without prejudice to subsequent court proceedings) within one month after denial by the Chief of the Mines Service. (Art. 27.)

Duration.- A prospecting permit is valid for one year from date of delivery and may be renewed two or more times for additional periods of one year each, even though it may have changed hands. A fee is charged for each renewal, and application therefor must be addressed to the Chief of the Mines Service before the expiration of the existing permit. (Art. 28.)

A prospecting permit becomes null and void when it expires and unless a concession has been applied for the area becomes open again for prospecting. The former title holder, however, can not obtain a new permit covering the same area or any part thereof within one year after his old permit expires. (Art. 29.)

Disposal of product.- Prospecting work which degenerates into exploitation is forbidden and subject to punishment by fine and imprisonment as illegal mining. (Art. 32.) Nevertheless, a prospector may freely dispose of concessionable minerals produced by his labors subject to the payment of taxes and royalties for substances of like character. He also must notify the Chief of the Mines Service who furnishes a certificate of permission to ship. This authority is valid for one year and may be renewed. (Art. 31.)

CONCESSIONS

General.- Any prospecting permit which has not expired carries with it the right to obtain a concession. (Art. 33.) Application therefor is addressed to the Chief of the Mines Service and contains the same general information as the application for a prospecting permit, including a map (scale) 1:10,000) together with a description of the prospecting work and the nature and character of the deposit discovered. (Art. 36.) Promptly after the application is filed, the Mines Service proceeds to examine it (art. 39) and within 15 days the applicant must pay the requisite fee. (Art. 38.) If the application is apparently in due form, an investigation is started by the Mines Service, notice being given meanwhile in two issues of the Official Journal of the colony at least 15 days apart. (Art. 40.) While this inquiry is in progress, all oppositions may be formulated by third parties. Such opposition, if it relates to the ownership of the permit, should be instituted before the courts. (Art. 41.) All other opposition is addressed to the Chief of the Mines Service and to be effective must be presented during the inquiry. (Art. 42.) In the absence of opposition from other interested parties, the Governor is authorized to reject the claim for a concession only for fundamental irregularities in the application or title which the applicant does not clear up. (Art. 44.) If the prospecting permit expires before the application for a concession for the same area has been acted upon, the permit is automatically extended and the applicant may also exploit the mine under provisional title. (Art. 47.)

Area.- The unit area of a concession is a rectangle oriented due north and south with the short side at least one-fourth as long as the long side. The configuration may be modified, however, by special ruling (derogation) of the Chief of the Mines Service in cases where such procedure is deemed necessary. The area shall not be more than 2,500 hectares nor less than 100 hectares for mineral fuels; for other substances the upper and lower limits are 2,500 hectares and 5 hectares, respectively. All the area applied for must be within the area covered by the prospecting permit. (Art. 33.) Fractional areas between lands already covered by concessions or prospecting permits of less than the prescribed dimensions may be made the object of separate concessions or be annexed to adjoining concessions subject, of course, to prior application therefor. (Art. 34.)

Duration.- The period of the concession is 75 years and it may be renewed by the Governor for 25 years more, provided the property has been exploited sufficiently actively. Application for renewal should be addressed to the Governor at least five years before the original concession expires. (Art. 46.)

Annulment.- A concession may be cancelled at any time in favor of the owner of a valid prospecting permit or concession upon proof of prior registration of an application for a prospecting permit. (Art. 45.) Failure to pay taxes and other charges also may result in forfeiture. Provision is made for voluntary renunciation of all or part of a concession. (Art. 48.) Under certain conditions the former concessionnaire may salvage his tools and equipment, but buildings on the ceded area become the property of the Colony. (Art. 49.)

RENTS AND WORKING REQUIREMENTS

All concessions are subject to the payment of local taxes annually and an additional tax may be levied after the expiration of five years if the annual production fails to come up to a specified minimum. In the case of nickel, copper, iron, manganese, or gypsum ore, this minimum is 1 metric ton per hectare per year, and in the case of chrome ore and cobalt, it is even less; for other minerals there is a working requirement equivalent to at least four man-days per year per hectare. (Art. 52.) In the case of group developments work done on one concession may be credited to others in a unit group. (Art. 53.)

RELATIONS WITH LANDOWNERS AND OTHER MINERS

Walled enclosures, streams, and gardens can not be occupied without formal consent of the landowner, and no mine openings can be made without the latter's permission within 50 meters of a dwelling or adjoining land. (Art. 57.) The laws and customs respecting graves must also be observed. (Art. 58.)

On the free public domain, a concessionnaire may freely occupy within the limits of his concession all land necessary for his prospecting, mining, and the mechanical preparation of the product and also for ditches, canals, and communications, as well as markers and monuments -- subject to approval of the Governor. He may use any water not already utilized and cut timber necessary for his mining operations. Moreover, he has a prior right to acquire the surface rights for the same area. (Art. 59.)

Private lands can be occupied for mining purposes by authorization of the Governor but except in emergencies not until the amount of indemnity has been fixed. The indemnity is double the (normal) net revenue from the land occupied and is payable annually in advance. When the occupation will last for more than one year or when it renders the land unfit for its original use, the proprietor may insist upon purchase at double the value of the land before it was occupied. (Art. 60.) If the value of the land is not restored to its original use, dispute as to the appropriate amount of damages may be carried to the courts. (Art. 62.) Rights of way and other privileges outside of the concession are discussed in arts. 63 to 65. Damage to adjoining mines must be recompensed. (Art. 67.)

Holders of prospecting permits have similar rights to those of concessionnaires except with respect to the acquiring ownership of surface rights. (Art. 76.)

MINE INSPECTION

The usual provisions in French colonial law for supervision of mining operations by the Mines Service to assure health and safety of workers and protection of the public interest are contained in arts. 69-75.

JURISDICTION AND PENALTIES

Appeal from decisions of the Governor may be made before the Council of the State. (Art. 77.) Cases involving overlap of claims are decided by the courts, advice from the Mines Service being entered as expert testimony. (Art. 78.) Charges of violations of the mining law may be brought by the judiciary police, agents of the Mines Service, and all other agents commissioned for this purpose by the Governor. Such charges constitute proof in the absence of proof to the contrary, and opposition thereto must be entered within 30 days. (Art. 79.)

A fine of from 1,000 to 25,000 francs <u>and</u> imprisonment for from three months to three years are provided for illegal exploitation of precious stones or precious metals, and the product is confiscated. (Art. 80.)

A fine of from 100 to 1,000 francs or imprisonment for from 15 days to 2 years, or both, is provided for (1) willful destruction of or tampering with claim monuments, (2) falsification as to date of a prospecting permit, (3) false declaration of identity or misrepresentation regarding other essential matters in order to obtain a prospecting permit. (Art. 81.)

Minor infractions of the mining code are punishable by fines of lesser amount and imprisonment up to one year also may be ordered in case of illegal exploitation of deposits of other than precious metals or stones. (Arts. 82 and 83.)

For a second offense the maximum fine and imprisonment must be imposed and it may be doubled. (Art. 84.)

Those who have been imprisoned for infractions of the mining code become ineligible to obtain prospecting permits or concessions for three years and existing permits may not be renewed for a similar period. (Art. 86.)

MISCELLANEOUS PROVISIONS

The Governor with the advice of the Mines Service may promulgate regulations necessary for carrying out the provisions of the mining law. (Art. 95.)

The Mines Service (as organized under the Decree of August 5, 1910) is charged with the administration of the mining laws. (Art. 96.)

For the public interest the Governor in Privy Council may issue an order suspending for two years the right to obtain prospecting permits in designated regions. (Art. 97.) He may also requisition for Government use

any mineral produced, subject to suitable payment (which may be settled by the courts). (Art. 98.) He is further empowered to prohibit fusion of two or more mines under the ownership of the same individual or corporation, if such fusion appears contrary to the public interest. (Art. 99.)

With respect to penitentiary lands the provisions of the mining laws are applicable only as directed by the competent administrative authorities and with such reservations as are adjudged necessary. (Art. 100.)

Notice of the institution of prospecting permits or concessions, as well as all changes, court decisions, etc., are registered in accordance with the laws relating to real estate. Applications for and deliveries of prospecting permits and concessions and virtually all official business relating thereto are advertized in the Official Journal of the Colony.

Transfer of a prospecting permit or concession must cover the entire area. It may not be divided, although a partial lien may be executed after special authorization.

Notice of the official domicile of title holders or agents in Noumea should be furnished to the Mines Service. (Art. 14.)

All applications and accessory documents addressed to the administration must be in French and signed in French characters. All other documents must be either in French or accompanied by a French translation. (Art. 15.)

Employes of the Mines Service or Topographical Survey, in active service or on leave, are forbidden to take any interest, direct or indirect, in prospecting or mining in the Colony. Other Government employes stationed in New Caledonia are forbidden to take a direct interest. (Art. 16.)

The exploitation of mines is considered an act of commerce. (Art. 17.)

OIL AND GAS

The provisions of the general mining law are all applicable to substances of the fourth category except as provided for in articles 88 to 93 (and subsequent legislation).

Prospecting permits are issued for <u>four</u> years and may be renewed for two periods of two years each (art. 88) but such renewal is contingent upon the performance of certain minimum working requirements. (Arts. 89 and 90.)

Concessions for oil or gas are for 40 years and may be renewed under the same conditions as concessions for other minerals. The minimum output required (after five years) per hectare is 300 liters for hydrocarbons and 300 kilograms for asphalt or bituminous shale. (Art. 92.)

Under date of October 9, 1929, a special decree authorized the Governor, with the advice of the Mines Service, to set aside territories as oil and gas reserves in which prospecting or mining rights for minerals of this group (fourth category) may be granted only by virtue of a decree issued by the Governor with the advice of the French Committee of Public Works for the Colonies.

FRENCH ESTABLISHMENTS OF OCEANIA

GENERAL SUMMARY

French Oceania comprises a number of groups of islands scattered over a roughly square area in the central Pacific some 2,000 miles due east of New Caledonia. It includes the Marquezas Islands, the Tuamoto Archipelago, the Leeward Islands, the Gambier group, the Society Islands (notably the large island of Tahiti) and Tubuai or Austral Islands, together with several small and more or less isolated islands. The Governor and an Administrative Council have their seat at Papeete, Tahiti.

The New Hebrides group is not included in the French Establishments of Oceania nor in the Colony of New Caledonia but is under Anglo-French control. Except for sulphur (of volcanic origin), its importance as a source of commercial minerals is apparently negligible.

French Oceania is composed essentially of volcanic rocks and coral. From the mining standpoint, the only development is the moderately extensive phosphate production on Makatea Island of the Tuamotu group. Except for a little red ocher which is used locally as a pigment, and lignite which is intercalated in basaltic tuff at Rapa (Tubuai Islands), other economic minerals seem to be absent.

The mining law of 1810 of the mother country was applied to French Oceania by virtue of a pronouncement from Tahiti in 1874, but as it proved unsuitable the revised legislation for New Caledonia with necessary modifications was promulgated as the mining law of these Establishments by the decree of October 17, 1917, which was publsihed in the Official Journal of the French Republic of October 25, 1917 (vol. 49, No. 290, pp. 8495-8500).

As in other colonies of France, phosphates are classed as concessionable minerals (i.e., "mines") which are grouped in the same categories as in New Caledonia except that oil and gas are not separately provided for. Prospecting permits are issued for two years and may be renewed twice for one year at a time. Concessions are obtainable by holders of valid prospecting permits and there is a further provision for the issue of a concession by public adjudication with respect to certain areas designated by the Governor.

The area of a prospecting claim is defined as in New Caledonia. For a concession the area may be from 100 to 2,500 hectares for mineral fuels or from 5 to 2,000 hectares for other substances, the latter being a slight reduction from the 2,500 maximum allowed in New Caledonia. Under a later decree (February 23, 1918) relative to small islands, only one prospecting permit, which shall cover the entire island, may be issued for a single category of minerals on an island containing less than 5,000 hectares; the form of a concession can likewise be modified to conform with the outlines of a small island.

The general decree of February 27, 1924 -- requiring personal permits (authorization from the Governor) for all those engaged in mining or prospecting -- and other provisions that virtually assure French political control of corporations are applicable in French Oceania as well as in New Caledonia and other overseas possessions of France.

I.C. 6712
May, 1933.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

PORTABLE ELECTRIC LAMPS FOR ANIMAL HAULAGE IN ALABAMA[1]

By F. E. Cash[2] and C. E. Saxon[3]

PURPOSE OF REPORT

Adequate lighting for all mining operations would unquestionably tend to promote both safety and efficiency; this would be true not only for men but also for the animals that man impresses into service in mines. The Markeeta Coal Co. and the Alabama Fuel & Iron Co., both operating mines in Alabama, have found that equipping mine mules with portable electric lights is more efficient, much more humane, and far more safe for both men and animals than the more or less usual method of forcing the animal to do its work essentially in darkness.

This report is intended to give the mining industry data that will aid in trying to bring about safe and efficient operation of mines.

ACKNOWLEDGMENTS

The authors desire to thank Charles F. DeBardeleben, president, Fred R. Bell, general manager, Hewett Smith, C. H. Shepherd, and R. A. Sansing, superintendents for the Alabama Fuel & Iron Co.; and Charles F. DeBardeleben, Jr., president, and H. M. House, superintendent for the Markeeta Coal Co., for their cooperation in furnishing the data for this paper.

EXPERIENCE OF TWO COMPANIES

The eight mines operated by the Alabama Fuel & Iron Co. and one operated by the Markeeta Coal Co. in Jefferson and St. Clair Counties, Ala., are the only mines in Alabama where mules are provided with portable electric lights, and there are few if any mines in other States in which this system is in use.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6712."

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These two companies are convinced that equipping all mine mules with good lights decreases underground haulage cost by animals by prolonging the useful life of the animals, increasing their efficiency, lessening haulage wrecks and delays on haulage, and increasing production. Last but not least of the advantages would be the more humane treatment of the animals.

Mining Conditions

The nine mines are developed by slopes along the dip of the coal bed, with double-room entries on 200- to 300-foot centers turned right and left off the slopes and with approximately 1 percent grade in favor of the loads. Rooms are driven up or diagonally up the pitch from these entries.

A surface hoist delivers the empties from the surface to the entry sidetracks at the slope and pulls the loads to the surface. From these sidetracks, mules pull one to four wooden cars, each of which holds $1\frac{1}{2}$ to $2\frac{1}{2}$ tons of coal, deliver the empties to the rooms and entry faces, and pull the loads to the slope sidetracks. The mule haul varies from 100 to 3,000 feet; there are no locomotives or trolley wires in any of these nine mines, seven of which are rated as gassy and two as nongassy by the Alabama State Mining Department.

All employees in eight of the mines use permissible electric cap lamps; in one open lights are used.

All stables are on the surface and the mules are taken into the mines at the beginning of the working shift and brought out at the end of it; this is the usual practice at drift or gently dipping slope mines, and it is beneficial to the animals, lessens fouling of the mine air, reduces stable attendance, and eliminates a considerable underground fire risk from flammable materials.

HAULAGE IN INDIVIDUAL MINES

Markeeta Mine

The Markeeta Coal Co. operates one mine which produces about 375 tons of coal per day with eight mules engaged in haulage; the mules average 47 tons of coal hauled per animal per day. The mine was opened in July 1928, and electric mule lamps were put into use on March 1, 1932; prior to installing the mule lights, three mules were killed and several injured, but since the mule lights have been in use no animal has been killed or injured in the mine.

Acmar Mines

The Alabama Fuel & Iron Co. is operating three mines on the Acmar division (No. 2, No. 5, and No. 6), producing about 2,500 tons of coal per day with 50 mules; every mule, therefore, hauls an average of about 50 tons per day. During 1927, two electric mule lamps were put into use as an experiment and this number was gradually increased until on January 1, 1932, all of the mules employed in these mines were equipped with lights; before installing

mule lights, about nine mules per year were killed in the mines of the Acmar division, and several times as many were injured. During 1932 only one mule was killed in these three mines, a decided improvement over the record before the mule lights were adopted.

Margaret Mines

The Alabama Fuel & Iron Co. is operating two mines in the Margaret division (No. 3 and No. 5), producing about 1,800 tons of coal per day with haulage done by 30 mules; hence every mule pulls an average of about 60 tons per day. This division was completely equipped with mule lights on January 1, 1932; during 1932 but one mule was killed in these mines, while previously two or more mules were killed each year.

Overton Mines

The Alabama Fuel & Iron Co. is operating three mines on the Overton division (No. 1, No. 2, and Shades Valley), producing about 1,200 tons per day with haulage done by 21 mules; hence each mule hauls an average of about 57 tons per day. This division was fully equipped with electric mule lamps on January 1, 1932. Five mules were killed in 1929, 2 in 1930, and 2 in 1931. In 1932 these mines were responsible for the death of but one mule; this one was electrocuted when a car wreck pulled down a 220-volt, a.c., feeder cable.

Lamps Used

The Markeeta mine uses model C Wheat mule lamps carried in a leather case buckled to the mule's collar. The lamps are carried into the mine by the driver and are attached to the mule upon reaching the slope sidetrack. The total weight of the lamp, case, and strap, is 6 to 7 pounds.

In the Overton mines, Edison lamps are used and carried in a case similar to that used at the Markeeta mine, but at the Overton mines,the mules carry the collar and the lamp into and out of the mines; this method gives the mules the advantage of having their own light when traveling into and out of the mine.

The Margaret and Acmar divisions of the Alabama Fuel & Iron Co. use super-Wheat lamps carried in an aluminum case buckled to the mule collars; this case increases the weight 2 to 3 pounds. These lamps are also worn into and out of the mines by the mules.

These mule lamps differ from the permissible type worn by men in that they have no cord from the battery to the headpiece, the headpiece being fastened direct to the side of the battery case.

Mule Accident Experience

At the Markeeta mine, one mule was killed each year from 1929 to 1931. In 1932, after mule lamps had been provided, no mules were injured or killed. On this basis, the company has saved a mule a year by the installation of 10 mule lamps.

At the Acmar mines, the mules killed in 1930 cost $4,500, and $2,500 in 1931; in 1932, or after the mule lamps had been installed, the killing of mules in these mines cost but $100. This mule-lamp installation was begun with two lamps in 1927 and completed with 65 lamps in December 1931.

At the Margaret mines, the mule-lamp installation, 32 in number, was completed in December 1931. Two mules were killed in 1929 and cost $315; two in 1930 cost $430; no mules were killed in 1931, but in 1932 the death of one mule cost $100.

At the Overton mines in 1929, five mules killed cost $525; in 1930, two mules cost $150; in 1931, two cost $300, and in 1932 one cost $55. This mule-lamp installation was completed in December 1931.

SUMMARY

The Markeeta Coal Co., working eight mules, averaged one death each year before using lights and none after they were installed.

The Alabama Fuel & Iron Co. completed the installation of 121 mule lamps for 101 mules worked in all divisions in January 1, 1932. Since that time it is considered a rule violation to work a mule without a good light; if a lamp fails or if the light gets dim, the rule is that the mule must not be worked until another light is procured.

During 1930 this company had 16 mules killed, 11 in 1931 and 3 in 1932. These mules were being killed largely by being run over by cars, by running into standing cars, or by falling and breaking legs. Seldom are the mules killed outright by these causes, but a mule with a broken leg must be killed (as is the case with any large animal), hauled out of the mine, and buried.

The total cost of mules killed in 1930 was $5,080, $2,800 in 1931 and only $255 in 1932. These are the direct costs only and do not include numerous incidentals such as reduced tonnage or cost of removing dead mules.

The companies attribute the decrease in number of mules killed and the money saved during 1932 over previous years largely to the use of portable electric lights on all mules.

In addition to this saving in direct costs, the companies found during 1932 (or the first year of complete installation) that the mules not only get hurt less often and less severely, but they travel faster, pull more coal because they can see, and are often able to protect themselves as well as their drivers and in many cases the mine timbers or other property.

With two lights on a trip, one on the driver and the other on the mule, the driver is given a better chance to protect not only the mule but also to prevent himself from being injured. The mule not only benefits from additional light, but the drivers themselves suffer fewer accidents--and unquestionably many of the haulage accidents suffered by mule drivers are due to lack of light given to the animal.

CONCLUSION

If two coal-mining companies operating nine mines in Alabama by the use of 131 portable electric mule lights can save $2,000 or more per year in cost of mules killed, and in addition can haul more coal with considerably fewer wrecks and accidents to drivers, then all mining companies using animals for haulage should seriously consider the use of similar lighting equipment. Aside from the material gains to be achieved in reduced cost of animal supply or of compensation for injury or death of drivers, in lessened costs due to mine wrecks with possible tearing out of timber and letting down of loose roof, and in increased tonnage hauled per mule or per driver per day, any right-minded mine operator should consider the humanitarian features involved in the vastly increased protection to both animals and their drivers when adequate lights are given by mule-lamp installations.

I. C. 6713

May, 1933

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

ACCIDENT EXPERIENCE AND COSTS IN COLORADO METAL MINES

BY

E. H. Denny and E. A. Anundsen

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

ACCIDENT EXPERIENCE AND COSTS IN COLORADO METAL MINES[1]

By E. H. Denny[2] and E. A. Anundsen[3]

The prevalence of mine accidents in metal and coal mining in the various States has been responsible for detailed studies of accident causes and costs by the United States Bureau of Mines. Several reports[4] of the Bureau have recently appeared that deal with the cost of accidents. In addition, the Bureau publishes complete statistics yearly on the fatal accidents occurring in metal mines, coal mines, and quarries, also giving much data on nonfatal accidents.

It is felt that any figures presenting the true direct compensation and medical costs of mine accidents will serve to emphasize to mine operator and miner the economic as well as the humanitarian importance of the lessening of accidents. The present cost of compensation insurance is a considerable part of the labor cost; the tendency is toward legislative action with more liberal compensation payments for accidents to workers. Insurance companies in some cases are asking higher rates for compensation insurance, and state insurance funds are scrutinizing more carefully the ratio of premiums and losses for every branch of industry carried. Obviously, the most feasible means of keeping mine compensation insurance rates at their present level or reducing them is to decrease the number and severity of accidents.

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4 Ash, S. H., Accident Experience and Cost of Accidents in Washington Coal Mines: Inf. Circ. 6529, 1931, 18 pp.

Murray, A. L., and Harrington, D., Accident Experience of the Coal Mines of Utah for the Period 1918-1929: Inf. Circ. 6530, 1931, 26 pp.

Crawford, F. S., Medical Service, Accident Reports, Compensation, and Welfare at Iron Mines in the Lake Superior Region: Inf. Circ. 6567, 1932, 34 pp.

Fene, W. J., Accident Experience and Cost in Pennsylvania Anthracite and Bituminous Mines, 1926-1930: Inf. Circ. 6618, 1932, 29 pp.

Cash, F. E., Accident Experience and Cost in Tennessee Coal Mines: Inf. Circ. 6664, 1932, 8 pp.

Herbert, C. A., Ten Years of Fatal Accidents and Two Years of Accident Costs in Indiana Coal Mining: Inf. Circ. 6672, 1932, 12 pp.

This report discusses the cost and causes of Colorado metal-mine accidents in the period 1926 to 1930, inclusive. These accidents occurred in mines scattered over many counties. Numbers of them occurred at old reopened mines, first worked many years ago. Most of the operating mines are small in number of persons employed.

In 1930, John T. Joyce, commissioner of the Colorado Bureau of Mines, reported 361 mines or groups of mines, including 12 placers, located in 31 counties as operating. These operations gave 1,000,669 days of employment, which is equivalent to about 9 men at each operation working 290 days per year. Colorado reports more operating mines than do States with much larger metal production and many more men employed, as, for example, Nevada, Utah, Montana, Arizona, Michigan, and Minnesota. According to C. W. Henderson's chapter in Mineral Resources of the United States for 1930, Colorado metal mines in 1930 had production amounting to $13,265,701 in gold, silver, lead, and copper, and also more than $3,000,000 in tungsten, molybdenum, vanadium, iron, and manganese.

SUMMARY

Important facts brought out in this study of Colorado metal-mine accidents include the following:

1. The direct cost of 5,510 Colorado metal-mine accidents in a 5-year period in compensation, medical, hospital, funeral, and dental benefits was approximately $1,000,000, of which about 83 percent was compensation.

2. Each compensable case averages for compensation $464 and for medical attention $77.

3. The major compensation cost is that of fatalities, permanent total disabilities, and permanent partial disabilities. Fifteen permanent total disabilities have cost or probably will cost an average of over $18,000 each.

4. Compensable injuries involving no permanent disability cost only about $55 each in compensation but $54 in medical charges.

5. No-lost-time injuries, of which 1,949 were reported, cost about $7.50 each for medical and dental care.

6. A comparison of accident-frequency rates both as to fatal and nonfatal injuries shows that Colorado has one of the poorest records of the metal-producing States as far as fatalities are concerned and only a mediocre rate as far as nonfatal injuries are concerned.

7. Colorado metal mines employ a relatively small number of men per mine and many of the properties are old. Usually, the larger property can make a better safety record because of its organization and financial resources than can a small one; nevertheless, many small mines have enviable safety records and the writers believe that Colorado's metal mines can have them also.

8. The causes of accidents in Colorado metal mines are much the same as in other States, with falls of rock or ore, timber and hand tools, drilling, and haulage causing the larger percentages of accidents. Accident reports indicate in numerous cases that accidents could have been prevented, if reasonable precautions had been taken.

9. The approximately $1,000,000 direct cost of accidents for a 5-year period is probably only a fraction of the indirect and hidden costs of accidents which one authority puts at four times the direct cost.[5] This $1,000,000 direct cost does not include the cost of administration of the State fund nor that of other insurance carriers. The Colorado Industrial Commission states that the cost of writing insurance by the Colorado State Fund was 7.1 percent of the premiums received.

METHODS EMPLOYED IN ACCIDENT STUDY

Through cooperation of the Colorado Industrial Commission, engineers of the United States Bureau of Mines were given access to the individual reports of the Commission for all metal-mining accidents. The workman's compensation act of Colorado provides (section 30) that--

> Every employer shall keep a record of all injuries, fatal or otherwise, received by his employees in the course of their employment. Within 10 days after the occurrence of an accident resulting in personal injury, a report thereof shall be made, in writing by the employer to the Commission, upon forms prescribed by the Commission for that purpose.

These reports are in most cases fairly complete and give such information as the cause of the accident, length of disability if any, compensation paid, and in many cases the medical and hospital costs. The information in this report is based on examination of all individual accident reports for metal mines for the period 1926 to 1930, inclusive. It is believed that it furnishes a practically complete and accurate record of the number of accidents and compensation payments classified as to direct causes. In a few continuing cases of severe injuries it was necessary after consulting Commission officials to estimate the future compensation payments to be made. In cases of permanent total disability a life expectancy table (see p. 54, Workmen's Compensation Act of Colorado, Expectancy Table C.L. Sec. 6537) was used in computing compensation payments. Medical, hospital, funeral, and dental costs were available in all cases when paid by the state--that is, for companies insuring with the Colorado State fund and not insuring "exmedical." Self-insurers and companies insuring with commercial insurance companies in many cases do not report medical or hospital costs to the Commission. From the 3,055 cases for which medical costs were available an estimate of the total medical cost for all cases was made.

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5 Heinrich, H. W., Cost of Industrial Accidents to the State, the Employer, and the Man: Reprint of address before 17th annual meeting, Internat. Ind. Accident Boards and Commissions, Wilmington, Delaware, Sept. 22-26, 1930, p. 8.

As used in this report, the number of days lost from accidents involving no permanent partial or total disability is the number of days lost from the day of disability to time of return to work, including Sundays and holidays. If the man did not report back to the job the doctor's estimate of time lost from accident is used. For permanent partial and total disabilities and fatalities, the amount of awards varies in the several States, and to make figures comparable on days lost, the table for weighing industrial accident disabilities recommended by the Association of Industrial Accident Boards and Commissions and printed herein (Table 8) was used. Thus, a fatality or permanent total disability is rated at 6,000 days lost time and other permanently disabling injuries are rated proportionately.

The figures given include, for the period 1926 to 1930, inclusive, all metal-mine accidents in Colorado both on the surface and underground, a few accidents occurring in clay mines, and a very few gold-dredging accidents; they do not include any quarry, mill, smelter, coal-mine, or railroad-tunnel accidents. The compensable accidents listed are those for which the period of disability lasted longer than 10 days from the day the injured employee left work. They include a few cases in which no compensation was paid because of inability to locate the one-time injured man or in several fatal cases because of no dependents. The noncompensable cases include all cases reported where the injured person was disabled for 10 days or less; numerous "no time lost" accidents are included, most of which involve definite medical cost.

COST OF ACCIDENTS

From 1926 to 1930, inclusive, according to this study of the Industrial Commission's records, there were in and about Colorado metal mines 1,761 accidents involving a loss of more than 10 days time each and 3,749 accidents involving loss of time of 10 days or less each, a total of 5,510 accidents. For these 1,761 compensable accidents a total of $817,505.88 compensation has or will be paid and there was an estimated medical, hospital, and funeral cost, based on the actual cost of 787 compensable cases and 2,268 noncompensable cases of $135,052.42 for the more than 10-day lost time cases and of $32,452.79 for the noncompensable cases. This is a total direct cost of $985,011.09.

For each compensable accident, including fatalities, there was an average compensation cost of $464.23 and an average medical (including hospital and other charges) cost based on the above estimates of $76.69; for each noncompensable case there was an average medical cost of $8.66. The average direct cost of each of the 5,510 mine accidents reported (compensation, medical, hospital, funeral) was $178.77.

A large portion of the total compensation cost of Colorado metal mine accidents is due to fatalities, permanent total disabilities, and permanent partial disabilities. Thus, 96 fatalities cost in compensation $246,014.21 or an average of $2,562.65 per fatality. It is estimated that 15 permanent total disabilities will cost $271,009 in compensation, or that these men now totally blind or with broken backs will receive on the basis of $14.00 or

less per week an average of $18,067.23 each; 236 permanent partial disabilities, many involving such serious loss as that of an eye, cost $222,045 in compensation or an average of $940.87 each. The total compensation cost of the 347 fatalities and of permanent total and partial disabilities is $739,068, leaving a compensation cost of $78,438 for the 1,414 remaining compensable injuries involving no permanent disability. The average compensable case involving no permanent disability, therefore, costs only $55.47 in compensation. Assuming $14 weekly compensation payments with the 10-day waiting period for which no compensation is paid, the average injured man receiving compensation and having no permanent loss of members gets in compensation for 30 days of disability $55.47 of $1.85 per day. It is only fair to state, however, that some companies continue men on full or half salary or wages during the period of disability, such payments not being included in the above compensation records.

These 1,414 compensable injuries involving no permanent disability had an estimated medical cost, based on the actual recorded cost of 567 of these cases, of $75,995.49 or approximately $53.75 per injury. The doctors, therefore, fare nearly as well as the patients on compensation cases involving no permanent disability.

Of the 3,749 noncompensable cases recorded, 1,949 were reported as involving no loss in time. Nevertheless, there was an appreciable medical cost for these cases. Costs were available for 1,323 cases in total amount $9,886.72. This is an average of $7.47 medical cost per each no-lost-time case. A considerable amount of this cost was due to dental work necessitated by the injuries.

The time lost by the individual injured in mine accidents is of interest. In the 5,510 accidents reported there were 972,265 days lost or an average of 176.5 days per accident; 1,761 of these accidents were compensable and the average days lost for each compensable case was 547.5 days. It should be remembered that in these calculations a permanently disabling accident or a fatality is rated by the scale of time losses given in Table 8; thus, a loss of leg below the knee is equivalent to 3,000 days or a fatality 6,000 days. Of the accidents 3,749 were noncompensable, and the average time lost in these, including 1,949 no lost-time accidents, was 2.2 days. These data are given in detail in Tables 3 and 4.

CAUSES OF ACCIDENTS

The proximate causes of accidents according to the classification of the United States Bureau of Mines and the Colorado State Bureau of Mines are also given in Table 6 with the average loss of time from each cause. However, some comment on these proximate causes is worth while in connection with the more detailed information available in the Commission's individual reports. The term "proximate cause"[6] is used because often many factors contribute to an accident such as condition of surroundings or equipment, illumination or lack thereof, discipline, supervision, state of mind of the injured at time of accident, fatigue, ill health, lack of mental alertness, carelessness,

recklessness, loose clothing, and possibly other influences. Sometimes, one or more of these factors may be of more importance in injury causation than that ascribed under any of the usual headings.

Cause 1: Falls of rock or ore from roof or wall.- Accounts for 987 accidents. Here, as is usual in mines, is the largest cause of both serious and slight injuries and in these tabulations of 18 per cent of all Colorado mine accidents. In the Commission's reports appeared evidence in many cases of insufficient testing of roof, insufficient barring down of loose rock, insufficient supervision and discipline, insufficient timbering, and carelessness. Falls of rock while drilling were particularly frequent. In many cases protective hats or shoes would have minimized the injury.

Cause 2: Rock or ore while loading at working face.- Accounts for 388 accidents or 7 percent. Accidents from this cause included rocks rolling down muck pile, strains from lifting and shoveling, injuries to feet, hands, and eyes in loading operations.

Cause 3: Timber or hand tools.- Accounts for 712 accidents or 13 percent. Accidents included strains from handling timber and injuries of all sorts from hand tools. Sharp picks and axes and good handles for tools would have eliminated some of these accidents. Use of goggles would have prevented many accidents from flying particles of rock and steel.

Cause 4: Explosives.- Accounts for only 59 or 1 percent of all accidents, but over 2,731 days were lost per compensable explosive accident. Several of these accidents were fatal. Even more serious, some were cases of total loss of eyesight, only slightly atoned for by award of compensation for life. There was much evidence of careless practice in the handling of explosives and detonators and insufficient examination or working places for missed shots.

Cause 5: Haulage.- Accounts for 609 accidents or 11 percent of all accidents. Lack of clearance and poor track have contributed to some of these accidents.

Cause 6: Falling down chute, raise or stope.- Accounts for 219 accidents or 4 percent of all accidents. It is also noted that the severity rate (526 days lost per accident) is particularly high. Despite frequent assurances of mine officials and miners that they are so familiar with their traveling ways that accidents do not happen, it is evident from these figures that men do fall down chutes, raises, and stopes and receive serious injuries. Observations made in some mines lead to the conclusion that it is remarkable that many additional such accidents do not happen.

Cause 7: Run of ore from chute or pocket.- Accounts for 354 accidents or over 6 percent of all accidents. The subject of chute gates is thoroughly

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6 U. S. Bureau of Labor Statistics, Standardization of Industrial Accident Statistics: Bull. 276, 1920, 103 pp.

discussed in United States Bureau of Mines Information Circular 6495, entitled "Underground Chute Gates in Metal Mines" (Aug. 1931). In some cases better chute gates would prevent injury to men drawing rock or ore.

Cause 8: Drilling.- Accounts for 612 accidents or over 11 percent of all accidents. Accidents from drilling are frequent but less severe than numerous other causes of accidents. A frequent item in Colorado accident reports is "piece of rock in eye while drilling," and particularly while collaring a hole; the use of goggles would have prevented such accidents. Other frequent occurrences are strains while handling drill, and fall of drill or drill steel on person; in many cases use of "safety" shoes would prevent accidents of this type.

Cause 9: Electricity.- Accounts for only 9 underground accidents recorded as due to electricity. This low number is largely due to the present limited use of electricity in Colorado metal mines, and particularly the little use of trolley locomotives. In mines in other States, electrical contact hazard and fire hazard have taken their toll. All electrical installations should be made with a careful regard to fire and contact hazards.

Cause 10: Machinery.- Accounts for 66 accidents or a little over 1 percent of all accidents. The severity rate of such accidents is quite high and unquestionably proper guarding of machinery would have prevented many injuries from this cause.

Cause 12: Suffocation from natural gases.- Although few accidents (14) were recorded from asphyxiation during the 5-year period, yet Colorado has had over 40 mine fatalities from this cause. The importance of testing mine air, particularly when entering abandoned metal mines, with a candle or carbide lamp must be held in mind. Many abandoned mines may be filled with air mixtures deficient in oxygen, the breathing of which will suffocate the victim almost instantly.

Causes 14 and 26: Stepping on nails, underground and on the surface.- Accounts for 151 accidents and about 3,500 days total loss of time. The little upturned nail has cost over $2,600 in compensation and over $1,300 in doctors' bills; "good housekeeping" in and about the mine would have prevented most of these accidents.

Causes 16 to 20: Shaft accidents.- Accounts for 124 accidents. These accidents are relatively infrequent but severe, as might be expected in the case of breakage of cables, cage troubles, and objects falling down shafts; many of these accidents are preventable if readily available precautions are taken.

Causes 22 to 30a: Surface accidents.- Accounts for 909 or about 16 percent of all accidents. The high severity rate of accidents in connection with railroad cars and locomotives is noteworthy. Electricity (cause 28) also has an even higher severity rate due to several fatalities from electrical contact. Falls of person, hand tools, and handling material account for over half of the surface accidents.

Cause 30b: Other causes.- This miscellaneous classification includes fatalities from aerial tramways and from snowslides, also from burns received on the surface from handling carbide lamps.

ACCIDENT FREQUENCY AND SEVERITY RATES

Accident frequency and severity rates in Colorado metal mines have been calculated (see Table 7) from the lost-time accidents listed in the Colorado Industrial Commission reports and from the number of days of employment given in the Colorado State Bureau of Mines annual reports. Calculations show that over the 5-year period 1926 to 1930, inclusive, there was an annual average of 163 lost-time injuries, including 4.4 fatalities per thousand 300-day workers. In other words, approximately 159 of every 1,000 workers receive lost-time injuries each year and an average of 4.4 are killed. There is an average of 18.5 days lost time for each 1,000 hours or 125 days worked. According to the United States Bureau of Mines Bulletins 292 (1926), 310 (1929), 320 (1930), 342 (1931), and 362(1932), the relative rating of Colorado as to accident frequency compared with 23 other metal-producing States were as follows:

Year	Rank in fatality rates	Rank in lost-time injury rates
1930	23	17
1929	24	19
1928	17	9
1927	24	7
1926	9	7

The only comparison the writers have found as to metal-mine severity rates is that of 52 metal mines competing in the National Safety Competition of 1930; according to Bureau of Mines Report of Investigations 3126 (1931) their average severity rate was 5.3 days lost per 1,000 hours of exposure or less than one third of the Colorado rate.

COLORADO WORKMAN'S COMPENSATION ACT

Important features of the Workman's Compensation Act of Colorado as amended in 1929 and 1931 include:

1. Administration of the act by an industrial commission.

2. Every employer of four or more employees (not including private domestic servants and farm and ranch laborers) is presumed to have accepted the provisions of the compensation act unless he has filed a notice in writing with the commission to the contrary.

3. An employer may secure compensation for his employees by insuring in the State Compensation Insurance Fund, by insuring with any authorized stock or mutual corporation, or by procuring a self-insurance permit from the commission.

4. Every insurance carrier writing compensation insurance must write it at rates approved as adequate by the commission.

5. If an employer subject to the compensation act has allowed his insurance to terminate or has not complied with the insurance provisions of the act, an injured employee or his dependents if he is killed may claim compensation, and the amount of compensation or benefits is thereby increased 50 percent over that provided by the act.

6. Every employer is required to keep a record of all injuries, fatal or otherwise, received by his employees in the course of their employment and to report an accident to the commission within 10 days of its occurrence upon forms prescribed by the commission and containing such information as required by the commission.

7. Salaries and expenses of the State Compensation Fund are paid out of the earnings of this fund, but must in no case exceed 10 percent of the premiums written by the Fund during the preceding year.

8. Every employer must furnish such medical, surgical, nursing, and hospital treatment, medical, hospital and surgical supplies, crutches and apparatus as may be reasonably needed at the time of injury and thereafter during the disability, but not exceeding four months from the date of accident and $500 in value.

9. In case of death, the dependents receive 50 percent of the deceased employee's average weekly wage not to exceed a maximum of $14 per week or a minimum of $5 per week for a period not to exceed 6 years. Marriage or death of any dependent terminates the dependency, as does the reaching of age of 18 by a son or brother of the deceased. This amounts to a maximum death benefit of $4,375. Dependents not residents of the United States receive only one fourth of the amount provided for residents.

In case of temporary disability of more than 10 days duration, the employee receives 50 percent of his average weekly wages so long as such disability is total, not to exceed $14 per week or less than $5 per week unless his wages are less than $5 per week, in which case he receives the actual wage. No compensation is paid for the first 10 days of disability.

In case of permanent partial disability the injured employee receives in addition to compensation for temporary disability, compensation based on the importance of the member lost; thus for loss of 1 eye he receives 104 weeks compensation, for loss of leg at hip 208 weeks, and for loss of arm at shoulder 208 weeks.

In case of injury to the teeth of an employee he receives in addition to disability benefits dental treatment not to exceed $100 in value.

In case of permanent total disability, the award is 50 percent of the average weekly wage subject to maximum and minimum limitations already described, continuing to the death of the injured person.

10. Compensation provided under the compensation act is reduced from the wilful failure of the employee to use safety devices provided by the employer, or from wilful failure to obey any reasonable rule adopted by the employer for the safety of the employee, or where injury results from intoxication of the employee.

11. The present (Feb. 1933) State Fund insurance rate is $4.27 per $100 of pay roll for Colorado metal mines. A 30 percent dividend return was made in 1932. The rate (Oct. 31, 1932) of private insurance companies for Colorado metal mines is $7.70 per $100 of pay roll, this being increased from a previous rate of $6.84.

The complete text of the workmen's compensation act of Colorado, obtainable from the Colorado Industrial Commission, Denver, should be referred to for fuller information.

MINE-ACCIDENT PREVENTION

A brief discussion of accidents classified according to causes is given in the foregoing pages of this report. It is evident that the provision of adequate protective equipment is not sufficient to prevent all accidents, and that proper supervision and discipline from the top of the company organization down is an essential factor for safe operation. A glance at the description of the various types of accidents furnishes the conviction that proper supervision supported by effective disciplinary measures might have prevented many of these accidents.

The burden of accident prevention rests with the management, who must stand solidly behind the safety movement. In addition to providing adequate supervision and discipline the company should show its desire for safety by proper safeguarding of mine surroundings and equipment, periodic inspections of equipment and operations, and general cleanliness and orderliness--in other words, good housekeeping. In the larger mines the men can be made safety-minded by means of publicity, instruction, bulletins, safety committees with proper departmental representation, and safety meetings. Officials and men should be required to familiarize themselves with and comply with state and company safety rules and practices. In small operations men can still comply with state rules and familiarize themselves by reading and discussion of safe practices. The man in charge of a small operation, say less than 25 men, can act as a safety committee of one toward accident prevention.

The writers have studied the problem of accident prevention in Colorado metal mines not only from the reports on accidents made to the Colorado Industrial Commission, but also by examination of some 30 Colorado metal mines. Recommendations have been made to the recent annual meetings of the Colorado Metal Mine Association, and a summary of substandard and dangerous conditions noted was made in a publication entitled "Safety and Compensation" published jointly in 1931 by the Colorado Metal Mining Fund, the Colorado Industrial Commission, and the Colorado Bureau of Mines.

SUBSTANDARD CONDITIONS

Many of the substandard conditions published in the report mentioned still exist in some Colorado metal mines. They are, therefore, listed here as factors partly responsible for Colorado's high accident rate and factors which should be eliminated if this rate is to be substantially improved. They are as follows:

1. Miners were observed to delay or entirely neglect to take down loose rock in the back or hanging wall. Rock falls cause 18 percent of Colorado metal-mine accidents; the remedy is carefulness and discipline; the fault rests with the management as well as the miner.

2. Miners were observed to delay in cleaning down manways and raises after blasting or to neglect to do this work. Loose rock and debris over a man on a ladder is a double menace; this is again a matter of carefulness which should be enforced by proper supervision and discipline.

3. Ladderways traveled frequently by men were observed and traveled, extending 40, 60, and more feet vertically, or even leaning outward from the vertical, these ladderways having no intermediate platforms or landings or having inadequate platforms or landings. Doors are frequently not installed in sollars. Ladders were traveled having from one to four consecutive rungs missing; some laddors were insufficiently or insecurely fastened in place where ladder slippage would mean a probable fatal fall.

4. Winzes were observed without timbered collars.

5. Manways opening into the levels above were observed without protecting doors or covering.

6. Large abandoned stopes above traveled levels were observed with no protection afforded against possible loose rock. Abandoned stopes opening along traveled drifts were observed to be not railed off.

7. Open ore passes were observed to be not railed off or protected by grizzlies.

8. Use of insufficient platforms or of no platform on working floors of square-set stopes and in raises was observed. Instances of insufficient timbering and timbering of too light a nature were seen.

9. Dry drilling was noted in several mines; this is particularly dangerous in siliceous ground.

10. Men loading cars from chutes were seen taking unnecessary chances such as standing directly in front of the chute mouth while barring down "hung up" rock or ore.

11. Many chute gates were of a type awkward to handle and not providing sufficient control where the ore might hang up temporarily and then come with a sudden rush.

12. Explosive boxes were seen being opened with a pick; wooden tools should be used.

13. Careless and rough handling of explosives was seen.

14. Instances of storage of boxes of explosive and capped fuse within a few inches of the rail were observed. Capped fuse was frequently seen stored within 12 feet of boxes of explosion. Boxes of explosive were found under loose rock with the top of the box removed. Capped fuse and explosive were even found in the same box in one instance. Detonators and explosives should always be stored a safe distance apart, both of them in closed rigid containers.

15. Overhead 440-volt power lines were observed less than 5 feet 9 inches above the rail, hence affording hazard of electrocution. In several instances electrical installations and wiring were substandard with respect to insulation, suspension, and guarding against man contact.

16. Old timber, powder boxes, paper and other flammable debris were observed piled random in crosscuts and wide spots without regard to fire hazard. A small timber fire may make sufficient smoke and carbon monoxide to imperil every man in the mine.

17. Surface shaft houses and other mining buildings which were not fire-proofed were observed; many of these buildings were so located that a fire in them might imperil the lives of the men working in the mine. Because of connections underground and lack of fire doors, fires in buildings or abandoned properties in some cases might imperil the lives of men working in adjoining mines. In several cases fire doors were not provided at openings of working mines about which were wooden buildings, and in some cases the doors were in such location and of such construction that they might give only temporary protection before they were burned out.

18. In some cases an almost total lack of underground fire protection was noted and inadequate surface fire protection was also observed.

19. Mine locomotives were seen operating in some places where there was almost no man clearance and no refuge holes provided for men.

20. Much work is so scattered in old mines that men work with little supervision; this is particularly true of men working under the leasing system, who sometimes have little mining experiences.

21. Carbide is sometimes stored in too large quantities underground without regard to surroundings.

22. Men were found working without eye protection at tasks involving definite hazards to the eyes. It is not sufficient to provide goggles; their use must be enforced. At some mines hard hats were not worn and protective shoes were practically unknown.

23. Lack of accident records classified as to causes was reported in several instances.

24. Lack of safety organization or safety meetings was noted at several mines.

The foregoing is not a complete list of substandard practices and conditions observed; it is desired to call attention, however, to the fact that many of them can be corrected by the operator or miner at little or no expense.

ACKNOWLEDGMENT

The writers wish to acknowledge the helpful assistance received, both in compiling this report and in interpreting the data therein, from the Colorado Industrial Commission and its officials and employees, particularly Thomas Annear, chairman; Mr. Redding, manager of the State Compensation Fund; and Mr. Smith, secretary; and also from John T. Joyce, Colorado State Commissioner of Mines.

APPENDIX

The following tables (1 to 8) give detailed figures on the cost, frequency, and severity of Colorado metal-mine accidents. The main facts disclosed by these tables have been given in this report.

Table 1. - Compensable accidents in Colorado metal mines, 1926 - 1930, inclusive

Year	Compensable accidents	Days lost	Compensation cost, all accidents	Medical cases reported	Medical cost of reported cases	Estimated total medical cost based on all accidents	Estimated total compensation and medical cost
1926	357	162,511	$192,412.27	138	$10,564.32	-	-
1927	386	243,551	177,460.24	160	13,924.38	-	-
1928	310	176,248	144,911.15	134	12,176.85	-	-
1929	337	204,883	160,992.96	193	17,110.20	-	-
1930	371	176,914	141,729.26	162	12,242.21	-	-
Total	1,761	964,107	817,505.88	787	1/66,017.96	2/$135,052.42	$952,558.30
Average	352.2 per year	192,821 per year	464.23 per case	-	-	76.69 per case	540.92 per case

1/ Actual medical cost of 787 cases. 2/ Calculated by estimating costs of fatalities, permanent disabilities, and temporary lost-time disabilities separately. See Table 5.

Table 2. - Noncompensable accidents in Colorado metal mines, 1926 - 1930, inclusive

Year	Non-compensable accidents	All noncompensable accidents: Days lost	All noncompensable accidents: Medical cases reported	All noncompensable accidents: Medical cost reported cases	Estimated total medical cost	No-lost-time accidents: No lost-time accidents	No-lost-time accidents: Medical cases reported	No-lost-time accidents: Medical cost of reported cases	Estimated total medical cost
1926	619	1,568	383	$3,059.25	-	273	197	$1,420.10	-
1927	725	1,610	489	4,453.48	-	392	295	2,216.23	-
1928	724	1,581	465	3,531.00	-	370	263	1,697.75	-
1929	915	1,931	530	5,023.20	-	478	313	2,637.70	-
1930	766	1,468	401	3,565.75	-	436	255	1,914.95	-
Total	3,749	8,158	2,268	19,632.68	1/$32,452.79	1,949	1,323	9,886.73	2/$14,564.80
Average	749.8 per year	1,632 per year	-	-	8.66 per case	390 per year	-	-	7.44 per case

1/ Actual medical costs were available on 2,268 of the 3,749 cases reported; from these costs the total medical cost was calculated.

2/ Actual medical costs were available on 1,323 of the 1,949 cases reported; from these costs the total medical cost was calculated.

Table 3. - Compensable accidents in Colorado metal mines, 1926 - 1930, inclusive showing causes, costs, and days lost

Cause	Cases	Days lost	Average days lost	Compensation cost	Average compensation cost	Medical, funeral, and hospital cost of reported cases	Medical, funeral, and hospital cases reported
Underground:							
1. Fall of roof or ore from roof or wall	324	202,520	625.06	$171,323.80	$528.78	$13,319.58	140
2. Rock or ore while loading at working face	117	47,932	409.68	26,068.32	222.81	2,320.80	39
3. Timber or hand tools	132	40,793	309.04	34,778.87	263.48	5,410.90	68
4. Explosives	46	125,643	2,731.37	182,433.37	3,965.94	3,233.14	25
5. Haulage	243	71,817	295.54	43,023.33	177.05	5,987.38	100
6. Falling down chute, raise or stope	105	114,825	1,093.57	71,839.63	684.19	5,043.62	58
7. Run of ore from chute or pocket	114	48,314	423.81	31,361.32	275.10	3,755.28	37
8. Drilling	137	26,979	196.93	35,291.51	257.60	4,643.66	53
9. Electricity	3	143	47.67	200.57	66.86	100.00	1
10. Machinery (other than locomotives or drills)	32	22,684	708.88	14,411.80	450.37	1,881.90	19
12. Suffocation from natural gases	2	90	45.00	147.14	73.57	122.70	2
14. Stepping on nails	8	1,165	145.62	1,064.22	133.03	988.05	5
15a. Handling materials (other than rock or ore)	25	8,331	333.24	16,389.31	655.57	765.10	11
15b. Other causes	27	15,323	567.52	26,990.77	999.66	817.64	9
Underground totals and averages	1,315	726,559	552.52	655,323.96	498.35	48,389.75	567

Table 3. - Compensable accidents in Colorado metal mines, 1926 - 1930, inclusive, showing causes, costs, and days lost--Continued

Cause	Cases	Days lost	Average days lost	Compensation cost	Average compensation cost	Medical, funeral, and hospital cost of reported cases	Medical, funeral, and hospital cases reported
Shaft:							
16. Falling down shaft	2	49	24.50	$23.46	$11.73	$102.10	2
17. Objects falling down shaft	11	6,636	603.27	4,534.45	412.22	212.50	4
18. Breaking of cables	4	7,274	1,818.50	1,916.35	479.09	194.00	3
20. Skip, cage or bucket	34	49,378	1,452.29	29,637.02	871.68	2,782.30	23
21. Other causes	1	15	15.0	8.57	8.57	0	0
Shaft - Totals and averages ...	52	63,352	1,218.31	36,119.85	694.61	3,290.90	32
Surface:							
22. Mine cars, mine locomotive.....	43	9,077	211.09	6,265.15	145.70	702.40	18
23. Railway cars and locomotives...	6	6,175	1,029.17	3,969.70	661.62	193.00	2
24. Run or fall of ore in or from ore bins	6	1,376	229.33	1,102.79	183.80	268.00	3
25. Falls of persons	85	13,714	161.34	19,042.49	224.03	2,498.48	38
26. Stepping on nail	10	1,995	199.50	1,572.91	157.29	367.25	6
27. Hand tools, axes, bars, etc. ..	50	7,993	159.86	7,567.74	151.35	1,781.97	26
28. Electricity	13	23,421	1,801.62	11,026.80	848.22	714.00	7
29. Machinery	45	27,000	600.00	16,815.92	373.69	1,748.29	16
30a. Handling materials	65	7,111	109.40	7,151.56	110.02	1,537.90	29
30b. Other causes	69	76,181	1,104.07	51,294.39	743.40	4,343.02	41
Surface totals and averages ...	392	174,043	443.99	125,809.45	320.94	14,154.31	186
No cause listed	2	153	76.50	252.62	126.31	183.00	2
Grand totals and averages ...	1,761	964,107	547.48	817,505.88	464.23	66,017.96	787

Table 4. - Noncompensable accidents in Colorado metal mines, 1926- 1930, inclusive, showing causes, costs, and days lost

Cause	Cases	Days lost	Average days lost	No days lost time	Medical cases reported	Medical cost	Average medical cost
Underground:							
1. Fall of rock or ore from roof or wall .	663	1,467	2.21	322	427	$4,083.80	$9.56
2. Rock or ore while loading at working face	271	488	1.80	155	142	966.15	6.80
3. Timber or hand tools	580	1,066	1.84	319	357	2,783.10	7.80
4. Explosives	13	45	3.46	5	4	26.50	6.62
5. Haulage	366	1,047	2.86	156	200	1,541.75	7.71
6. Falling down chute, raise or stope	114	357	3.13	45	61	611.50	10.02
7. Run of ore from chute or pocket ...	240	588	2.45	107	159	1,480.25	9.31
8. Drilling	475	731	1.54	307	345	2,728.40	7.91
9. Electricity	6	19	3.17	1	3	31.00	10.33
10. Machinery (other than locomotives or drills)	34	74	2.18	16	13	108.00	8.31
12. Suffocation from natural gases	12	12	1.00	10	8	33.50	4.19
14. Stepping on nails .	92	210	2.28	51	59	470.50	7.97
15a. Handling materials (other than rock or ore)	72	209	2.90	35	37	305.90	8.27
15b. Other causes	88	209	2.38	46	49	458.00	9.35
Underground totals and averages	3,026	6,522	2.16	1,575	1,864	15,628.35	8.38
Shaft:							
16. Falling down shaft.	2	12	6.00	0	0		
17. Objects falling down shaft	23	53	2.30	15	15	232.00	15.47
18. Breaking of cables.	1	4	4.00	0			
20. Skip, cage, or bucket	40	95	2.38	18	20	172.00	8.60
21. Other causes	6	15	2.50	4	5	34.50	6.90
Shaft totals and averages	72	179	2.49	37	40	438.50	10.96

Table 4. - Noncompensable accidents in Colorado metal mines, 1923 - 1930, inclusive, showing causes, costs, and days lost--Continued

Cause	Cases	Days lost	Average days lost	No days lost time	Medical cases reported	Medical cost	Average medical cost
Surface:							
22. Mine cars, mine locomotives	51	191	3.75	17	34	462.50	13.60
23. Railway cars and locomotives	3	7	2.33	2	1	5.50	5.50
24. Run or fall of ore in or from ore bins	11	24	2.18	6	5	67.50	13.50
25. Falls of person ...	80	215	2.69	33	51	689.13	13.51
26. Stepping on nail ..	41	98	2.39	17	14	103.00	7.36
27. Hand tools, axes, bars, etc.	136	203	1.49	90	74	596.75	8.06
28. Electricity	14	31	2.21	4	9	99.50	11.06
29. Machinery	80	154	1.92	42	56	593.80	10.60
30a. Handling materials	170	324	1.91	96	82	631.20	7.70
30b. Other causes	61	192	3.15	29	35	280.95	8.03
Surface totals and averages	647	1,439	2.22	336	361	3,529.83	9.78
No cause listed	4	18	4.50	1	3	36.00	12.00
Grand total and averages	3,749	8,158	2.18	1,949	2,268	19,632.68	8.66

Table 5. - Colorado metal-mine accidents, 1926 - 1930, inclusive, showing cost of fatal and permanent partial and total disability accidents

Year	Fatalities	Compensation cost	Medical, hospital, and funeral cost	Total cost	Permanent total disabilities	Compensation cost	Estimated medical and hospital cost of all cases	Estimated total cost
1926	13	$36,015.11	$3,210.10	$39,225.21	5	$105,903.50	-	-
1927	27	56,600.85	3,335.45	59,936.30	3	56,916.00	-	-
1928	17	43,310.08	2,379.00	45,689.08	3	51,980.58	-	-
1929	23	69,993.72	2,607.44	72,601.16	2	24,344.42	-	-
1930	16	40,094.45	1,974.26	42,068.71	2	31,864.00	-	-
Total 5 years	96	246,014.21	13,506.25	259,520.46	15	271,008.50	$4,644.56	$275,653.06
Average	19.2 per year	2,562.65 per case	140.69 per case	2,703.34 per case	3 per year	18,067.23 per case	309.64 per case	18,376.87 per case

Table 5. - Colorado metal-mine accidents, 1926 - 1930, inclusive, showing cost of fatal and permanent partial and total disability accidents--Continued

Year	Permanent partial disabilities	Compensation cost	Estimated medical and hospital cost of all cases	Estimated total cost	Grand total	
					Cases	Estimated grand total cost
1926	41	$35,735.80	-	-	59	-
1927	45	47,279.54	-	-	75	-
1928	51	38,745.95	-	-	71	-
1929	43	50,730.13	-	-	68	-
1930	56	49,554.05	-	-	74	-
Total 5 years	236	222,045.47	$40,906.12	$262,951.59	1/ 347	$798,125.11
Average	47.2 per year	940.87 per case	173.33 per case	1,114.20 per case	69.4 per year	2,300.07 per case

1/ Actual medical costs were available on 220 of the 347 cases; from these costs the total cost was figured.

Table 6. - Colorado metal-mine accidents, 1926 - 1930, inclusive showing all accidents by cause, percentage, and days lost

Cause	Cases	Per cent of total	Days lost	Average days lost
1. Fall of roof or ore from roof or wall	987	17.92	203,987	206.7
2. Rock or ore while loading at working face	388	7.04	48,420	124.8
3. Hand tools	712	12.93	41,859	58.8
4. Explosives	59	1.07	125,688	2,130.3
5. Haulage	609	11.05	72,864	119.6
6. Falling down chute raise or stope	219	3.97	115.182	525.9
7. Run of ore from chute or pocket	354	6.42	48,902	138.1
8. Drilling	612	11.11	27,710	45.3
9. Electricity	9	0.16	162	18.0
10. Machinery (other than locomotives or drills)	66	1.20	22,758	344.8
12. Suffocation from natural gases	14	0.25	102	7.3
14. Stepping on nail	100	1.81	1,375	13.8
15a. Handling materials (other than rock or ore)	97	1.76	8,540	88.0
15b. Other causes	115	2.09	15,532	135.1
16. Falling down shaft	4	0.07	61	15.2
17. Objects falling down shaft ..	34	0.62	6,689	196.7
18. Breaking of cables	5	0.09	7,278	1,455.6
20. Skip, cage, or bucket	74	1.34	49,473	668.6
21. Other causes	7	0.13	30	4.3
22. Mine cars, mine locomotives .	94	1.71	9,268	98.6
23. Railway cars and locomotives.	9	0.16	6,182	686.9
24. Run or fall of ore in or from ore bins	17	0.31	1,400	82.4
25. Falls of persons	165	2.99	13,929	84.4
26. Stepping on nail	51	0.93	2,093	41.0
27. Hand tools, axes, bars, etc..	186	3.38	8,196	44.1
28. Electricity	27	0.49	23,452	868.6
29. Machinery	125	2.27	27,154	217.2
30a. Handling material	235	4.26	7,435	31.6
30b. Other causes	130	2.36	76,373	587.5
No cause listed	6	0.11	171	28.5
Total	5,510	100.00	972,265	176.5

Table 7. - Accident frequency and severity rates in Colorado metal mines, 1926 - 1930, inclusive

Year	Days employment 1/	Fatal accidents	Permanent total disabilities	Permanent partial disabilities	Temporary disabilities: Over 10 days lost time	Temporary disabilities: One to 10 days lost time	No-lost-time accidents 2/	Frequency rate: Killed per thousand 300-day workers	Frequency rate: Lost-time nonfatal injuries per thousand 300-day workers	Total lost-time accidents per thousand 300-day workers	Total all accidents per thousand 300-day workers	Severity rate: Days lost from accidents	Severity rate: Days lost per 1,000 hours exposure 3/
1926	1,269,246	13	5	41	298	346	273	3.07	163.1	166.2	230.7	164,079	16.2
1927	1,210,315	27	3	45	311	333	392	6.69	171.5	178.2	275.4	245,161	25.3
1928	1,607,723	17	3	51	239	354	370	3.18	121.0	124.2	193.4	177,829	13.9
1929	1,467,814	23	2	43	269	437	478	4.70	153.6	158.3	255.9	206,814	17.6
1930	1,000,669	16	2	56	297	330	436	4.80	205.4	210.2	340.9	178,382	22.3
5-yr. Total	6,551,767	96	15	236	1,414	1,800	1,949	4.40	158.7	163.1	252.3	972,265	18.5

1/ Annual reports of State Metal Mine Commissioner, but deductions made in 1926 and 1927 for Moffat (railroad) tunnel.
2/ Not figured into frequency rate.
3/ Each working day estimated as 8 hours.

Table 8. - Scale of time losses for weighing industrial accident disabilities so as to show severity of accidents, according to the U. S. Bureau of Mines

Nature of injury	Degree of disability in per cent of permanent total disability	Days lost
Death	100	6,000
Permanent total disability	100	6,000
Arm above elbow, dismemberment	75	4,500
Arm at or below elbow, dismemberment	60	3,600
Hand, dismemberment	50	3,000
Thumb, any permanent disability of	10	600
Any 1 finger, any permanent disability of	5	300
2 fingers, any permanent disability of	$12\frac{1}{2}$	750
3 fingers, any permanent disability of	20	1,200
4 fingers, any permanent disability of	30	1,800
Thumb and 1 finger, any permanent disability of	20	1,200
Thumb and 2 fingers, any permanent disability of	25	1,500
Thumb and 3 fingers, any permanent disability of	33-1/3	2,000
Thumb and 4 fingers, any permanent disability of	40	2,400
Leg above knee, dismemberment	75	4,500
Leg at or below knee, dismemberment	50	3,000
Foot dismemberment	40	2,400
Great toe, or any 2 or more toes, any permanent disability of	5	300
1 toe, other than great toe, any permanent disability of	0	-----
1 eye, loss of sight	30	1,800
Both eyes, loss of sight	100	6,000
1 ear, loss of hearing	10	600
Both ears, loss of hearing	50	3,000

I. C. 6714

May, 1933

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

FLOTATION PROCESSES FOR CLEANING FINE COAL

This paper presents the results of work done under a cooperative agreement between the Northwest Experiment Station, of the United States Bureau of Mines, and the University of Washington, Seattle, Wash.

BY

H. F. YANCEY AND J. A. TAYLOR

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

FLOTATION PROCESSES FOR CLEANING FINE COAL[1]

By H. F. Yancey[2] and J. A. Taylor[3]

CONTENTS

INTRODUCTION

In this information circular is presented a summary of the more important articles that have appeared in the literature pertaining to the experimental work and to the commercial operation of the froth-flotation and Trent processes as applied to the cleaning of fine coal.

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1 The Bureau of Mines will welcome the reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6714."

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The chief utility of these processes from the standpoint of present washery practice lies in the treatment of the fine coal normally present in the feed and of that produced by degradation during the washing operation. Such treatment may be applied most advantageously to the sludge and slime or to the fine coal removed from the feed before treatment. The coal subjected to these processes must be of fine size: for the Trent process through 65 mesh or finer, and for the froth-flotation process through 1/10 inch; therefore, these processes can not compete with present washing machines, except where the coal is originally fine or where exceptionally fine crushing is required to break the coal and refuse apart. Crushed middlings would fall in this class. Application may be made where the fines in a coal are high in ash or where a high-ash sludge results from the disintegration of clay during wet washing.

FROTH-FLOTATION PROCESS

The importance of froth flotation in Europe as a means of cleaning coal is shown by the tonnage treated. As early as 1925 about a million tons of coal were treated by this process.[4] Table 1 gives the amounts treated in the various European countries for the year 1925.

Table 1.- Coal flotation in Europe for the year 1925, recorded by Berthelot

	Tons (metric)
England	275,000
Germany	260,000
Spain	200,000
Holland	100,000
Belgium	80,000
Total	915,000

In the boom year of 1927, 932,036 tons of coal were treated by froth flotation in Great Britain. Since that time, the tonnage treated has declined. Table 2 shows recent coal cleaning figures for Great Britain. The total amount of coal that was cleaned since 1927 has increased steadily, while the coal cleaned by froth flotation decreased until 1930; in that year, the amount of coal cleaned by flotation was almost twice the amount for 1929, which seems to indicate a comeback for the process in the British Isles.

4 Berthelot, C., Washing of Coal by Flotation: Chim. et Ind., Special number, May, 1927, pp. 334-353.

Table 2.- Quantity of salable coal cleaned in Great Britain, 1927-1930 (long tons)[1]

Method	1927	1928	1929	1930
Wet washed	49,053,761	57,363,146	65,944,925	66,285,326
Dry cleaned	1,454,291	2,773,885	5,323,194	6,305,438
Froth flotation.	932,036	189,057	63,077	123,882
Total	51,440,088	60,326,088	71,333,196	72,714,646

1/ Annual reports of the Secretary for Mines, Great Britain.

Principle of the Process

The froth-flotation process has been used successfully for ore concentration for years. Applied to coal, the process, briefly described, involves the agitation of raw coal (all sizes up to 1/10 inch) with three to ten times its weight of water and a small quantity of suitable reagents (0.5 to 2.0 pounds per ton of raw coal). The reagent with air and water forms bubbles that selectively attach themselves to the coal particles. The coal particles are buoyed up and float at the surface, where they may be removed, while most of the refuse material is wetted and remains behind.

Early Investigations

Some of the earliest investigative work on coal flotation of which the results have been published was conducted by Bacon and Hamor[5] at the Mellon Institute of Industrial Research, Pittsburgh, Pa. These investigators found that the process could be used successfully for cleaning fine coal and predicted an important rôle for the process in the future. They discovered that a crushing rather than a rubbing or rolling action was to be desired whenever it was necessary to reduce the size of the raw coal to be treated.

As a result of a series of studies of the froth-flotation process made in 1920-1922 by the Bureau of Mines at its Northwest Experiment Station, Seattle, Wash., in cooperation with the University of Washington, and reported by Ralston and Yamada,[6] the following conclusions were reached:

> 1. Tests on a bony coking coal from the Wilkeson mine, at Wilkeson, Wash., have shown that the cleanest coal can be floated first, the bony coal next, and the ash last by ordinary froth flotation.

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5 Bacon, R. F., and Hamor, W. A., Problems in the Utilization of Fuels: Jour. Soc. Chem. Ind., vol. 38, 1919, p. 163.

6 Ralston, O. C., and Yamada, G., Froth-Flotation Tests of Bituminous Coking Coal: Chem. and Met. Eng., vol. 26, 1922, pp. 1081-1086.
Ralston, O. C., and Wickmann, A. P., The Froth Flotation of Coal: Chem. and Met. Eng., vol. 26, 1922, pp. 500-503.

2. Very thin oils or soluble-frothing agents work best.

3. The fine sizes of coal float the most easily, and the coal in the very fine slime will float to some extent without the use of any flotation oil whatever.

4. Owing to the tendency of fine sizes to float so easily, they are probably "overoiled" and produce dirty concentrate when enough oil is added to float the coarser sizes. Screen analyses of the flotation concentrate show increasing percentages of ash in the finer sizes.

5. To take due advantage of the above facts it is necessary to use a flow sheet in which roughing treatment is followed by cleaning of both concentrate and tailing from the roughing cell, discarding as tailing the material which will not float in any of the re-treating cells. Re-treatment of middling with original feed is not desirable except for the middling obtained in cleaning the rough concentrate.

6. By this method of treatment a clean concentrate can be obtained, the bony portion of the coal forms a middling which may have a market value, and the tailing is too low in coal to be of any value.

Early British investigators[7] describe the operation of a small 8-cell froth-flotation unit that operated successfully at the Skinningrove Iron Co. in England. Starting with raw coking coal having an ash content of 10 to 16 per cent, three products were produced; a high-grade coking coal containing 3 to 4 per cent ash, a steam coal containing 10 to 20 per cent ash, and a refuse product containing 70 to 80 per cent ash content. The coking coal when carbonized gave a super product, a porous but strong coke very economical to use in a blast furnace. According to these authors, the cost of running this flotation unit was practically the same as that for a jig-washing process. It was also predicted that the flotation process would be applicable only when the original coal is fine or when crushing is part of the normal treatment - that is, for coking, gas making, briquetting, coal-dust firing, colloidal fuel, and other similar uses. British investigators claim that the facility with which the various grades of coal are separated is the most striking feature of the process.

7 Bury, E., Broadbridge, W., and Hutchinson, A., Froth Flotation as Applied to the Washing of Industrial Coal: Trans. Inst. Min. Eng., vol. 60, 1920-21, pp. 243-253.
Bury, A. F., and Bicknell, A., Purification of Coal by Froth Flotation: Coll. Guard., vol. 121, 1921, p. 337.

A description of an early experimental flotation plant and results obtained at Aberman, England, under a joint agreement between Minerals Separation (Ltd.) and the Powell-Duffryn Co., gives the flow sheet used for treating a variety of raw materials.[8] Runs were made with fines from run-of-mine coking coal, steam-coal waste, and washery silt. The equipment of this experimental plant consisted of a crushing unit, a vibrating screen, a flotation machine, and an Oliver continuous-vacuum filter, with the necessary conveying machinery. The feed, crushed to pass 6 mesh, was passed by a chute to a small bin at the head of the flotation plant. A 9-box flotation machine was used. The feed was introduced with a suitable amount of water into the No. 1 agitation box by a screw conveyor. Because the presence of shale slime impeded the action of the Oliver filter and increased the ash content of the recovered coal, the froth produced in the first five boxes was re-treated in the last four. The flotation reagents were generally added in the first box. As a rule, 1 to 1-1/2 pounds of cresylic acid was required for each ton of raw feed. A by-product gas-scrubbing effluent was used as the frothing agent, eliminating most of the cost of reagents. The clean coal fell over the lip of the frothing boxes into the tank of the Oliver filter. It was picked up by the drum and dewatered to about 10 per cent moisture, discharged onto a belt conveyor, and loaded into trucks. The waste material passed out of the machine and was conveyed to the waste dump by a launder. Water from the filter could be reused, but that containing the refuse required clarification. The raw waste treated contained 50.4 per cent of ash and the recovered coal 13.1 per cent of ash. The yield of recovered coal was 37 per cent. The ash in the recovered coal could be reduced to 7.3 or 9.4 per cent if the yield were reduced to 17 or 27 per cent, respectively.

Jones[9] has given a thorough discussion and presented the results of tests of the froth flotation of coal and the problems encountered. He points out that small particles float more readily than large particles, so that a certain amount of overlapping is bound to occur, but this interferes to a limited (not practical) extent when several products are made. Trouble with pyrite going with coal is admitted. Fusain (charcoal) may be removed, if present in deleterious amounts, by adding a protective colloid such as starch or glue to the flotation pulp. Jones recommends the treatment of minus 1/2 or 1/4 inch feed in the flotation machine. Only the clean fine coal of flotation size floats. The tailing is passed over a screen with 1/12-inch openings and the coarse coal is recovered by tabling; fine screening of the feed is thereby eliminated.

In addition to the usual methods of dewatering, such as gravity drainage with admixture of coarse coal and the use of continuous filters, Jones gives one method, adapted from the briquetting process, which is novel. Water is displaced from the coal surfaces by oil, due to preferential wetting. One to two per cent of tar or tar oil (basis dry coal) is added and the mixture of floated coal, tabled coal, and water (usually 50 per cent water) is heated to 200° F. by live steam. The coal flocculates and is separated from the water on a coco-mat draining belt. It may be further dewatered by passing through a briquet press or by other pressure or suction methods. In 1925 the process was employed on a large scale in the north of England and was giving satisfactory results.

8 Jones, F. B., The Froth Flotation of Coal: Proc. South Wales Inst. Eng., vol. 37, 1921, p. 331; Abstract in Coll. Guard., vol. 122, 1921, p. 1002.
9 Jones, F. B., The Froth Flotation of Coal: Jour. Chem. Met. and Min. Soc. South Africa, vol. 26, 1925, pp. 125-138.

British Plants

After the successful work of Bury, Broadbridge, and Hutchinson in Great Britain, many plants were erected both in England and on the continent. Some of the flotation plants that began successful operation in Great Britain are as follows: The Powell-Duffryn Co. in the Alverdare Valley; the Skinningrove Iron Co. in Wales; the Ashington Coal Co. in Durham; and the Oughterside Colliery in Cumberland.

The froth-flotation plant with a capacity of 25 tons per hour for the preparation of coking coal from raw screenings passing 1-1/2-inch round-hole screen, in operation at the Oughterside colliery, Cumberland, England,[10] was described in 1924. The cost of crushing the coal to flotation size (1/10 inch) was found to be too high, so that it was crushed only to 1/4 inch, and a Plat-O table is used to recover the coal which is too coarse to be floated from the flotation refuse. The feed was screened at 1/4 inch with a Hummer vibrating screen and fed to the first box of the machine, which consisted of two mixing boxes and eight agitation and froth boxes. Water amounting to three and one-half times the weight of coal was added to the mixing box; and the reagents, comprising a mixture of refined cresylic acid (0.77 pound per ton of coal), the aerating agent, and petroleum-gas oil (0.37 pound per ton of coal), which gives the froth stability, were added later as required. The quality of the washed product was controlled by the amount of cresylic acid and gas oil that was added as reagents. The froth contained 50 per cent of water and was dewatered to about 18 per cent by means of an 8 by 8 foot Oliver continuous filter at the rate of 26 to 28 tons per hour. The filter tank was fed by gravity from the flotation machine. Brass-wire 36-mesh cloth was used as a filtering medium. A continuous centrifugal drier was first installed to dewater the froth but failed entirely on account of the high proportion of fine material present. The discards from the eighth cell were put onto a 20-mesh jigging screen 6 feet long by 3 feet wide. Monel metal screens were preferred to steel screens because of the rapid corrosion of steel. The undersize from the 20-mesh screen was discarded as waste while the oversize was sprayed before leaving the screen and discharged onto a Plat-O coal-washing table which had a capacity of 10 tons per hour. The amount of water required on the table was 2-1/2 times the weight of the coal treated. The coal from the table was divided into three products; a coking coal, a firing coal, and a refuse product. The coking coal from the table was then intimately mixed with the flotation concentrate and sent to the coke plant.

Table 3 gives a comprehensive set of results showing the distribution of the froth products for the first seven cells of the flotation machine at this plant. In addition to the results shown in Table 3, the authors state that a reduction of the phosphorus content of the coal from 0.040 to 0.0067 per cent took place in the flotation machine. The total froth had an average ash content of 5.3 per cent. The ash and sulphur contents, as well as the average size of the coal particle in the froth, increased very noticeably from the first to the last cells.

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10 Scoular, J. G., and Dunglinson, B., The Washing of Fine Coal by the Froth Flotation and Concentrating-Table Processes at Oughterside Colliery, Cumberland: Trans. Inst. Min. Eng., vol. 67, 1924, pp. 374-379. Abstracted in Coll. Guard., vol. 127, 1924, pp. 1061 and 1122; Gas World, vol. 80, 1924, coking section, pp. 53-57; Min. Mag., vol. 30, 1924, p. 379.

Table 3.- Samples from froth boxes taken simultaneously, according to Scoular and Dunglinson, per cent

Box No.	Ash	Sul-phur	On 1/10 inch	Dry screening 1/					Separation in 1.6 specific gravity					
									Float			Sink		
				1/10-inch to 10 mesh	10 to 20	20 to 60	60 to 100	Through 100	Weight	Ash	Sul-phur	Weight	Ash	Sul-phur
1	3.02	1.27	-	-	3.0	33.7	18.7	44.6	98.6	2.33	1.05	1.4	51.2	9.27
2	3.16	1.35	-	-	3.9	37.6	17.0	41.5	98.5	2.38	1.20	1.5	54.5	11.30
3	3.26	1.30	-	-	12.2	48.0	13.9	25.9	98.2	2.33	1.11	1.8	54.3	11.67
4	3.73	1.37	-	0.7	20.7	46.0	10.6	22.0	97.9	2.64	1.15	2.1	55.1	11.40
5	4.73	1.59	-	3.1	42.4	35.5	5.2	13.8	97.0	3.19	1.25	3.0	54.7	12.72
6	6.02	1.78	1.4	7.5	51.7	27.0	3.1	9.3	95.4	3.52	1.31	4.6	57.8	11.54
7	7.15	2.12	1.7	11.9	5.54	20.9	3.2	6.9	94.4	4.00	1.25	5.6	60.2	16.75

1/ Cement manufacturers' standard of England sieve scale.
Openings in inches: 10 mesh = 0.0754; 20 mesh = 0.0332; 60 mesh = 0.0114; 100 mesh = 0.0065; 200 mesh = 0.0023.

Average results for the operation of this plant for a month are shown in Table 4.

Table 4.- Average results of froth-flotation plant at Cumberland, England, according to Scoular and Dunglinson, per cent

Product	Ash	Float on 1.6 specific gravity		Sink in 1.6 specific gravity	
		Weight	Ash	Weight	Ash
Raw coal	21.58	75.4	4.77	24.6	73.10
Flotation coal, filter	5.30	97.0	3.90	3.0	50.70
Coking coal from table	5.00	100.0	5.00	-	-
Boiler coal from table	14.66	90.5	10.72	9.5	51.52
Refuse from 20-mesh screen.	75.80	-	-	100.0	75.80
Refuse from table	72.78	3.5	22.64	96.5	74.61

On the basis of moisture-free coal the average yields are as follows: 70.4 per cent of coking coal containing 5.25 per cent of ash, 7 per cent of boiler coal containing 14.66 per cent of ash, and 22.6 per cent of refuse containing 75.5 per cent of ash. As the coking coal recovered by flotation contained 5.3 per cent of ash and that from the table 5 per cent of ash and the mixture of the two contained 5.25 per cent of ash, it may be calculated that 83.3 per cent of the coking coal was furnished by flotation and 16.7 per cent by the table. It is pointed out that a slight disadvantage of the flotation process is the tendency of pyrite to float if the aeration is too strong and that this is likely to happen in the last few boxes of the machine when extra reagent is added to lift the coal of poorer quality. It was found that the froth from the last two or three boxes, when screened at 40 mesh, gave an undersize that was mainly pyrite.

The capital and operating costs of this plant for the year 1924 were as follows:

Capital cost of main items:	Pounds sterling	U. S. dollars[11]
Minerals Separation flotation machine	3,500	17,033
Oliver filter, including all pumps, air compressor, etc.	1,500	7,300
Hum-mer screen, with generator	600	2,920
Total capital cost	5,600	27,233

Operating costs:

(1) Crushing: The cost of crushing is practically the same as if the coal were washed in a jig washery and crushed after washing for coking.

(2) Screening, washing, drying, and disposal of products.

	Per ton, pence	Per ton, cents
(a) Labor: Three men and one youth with a charge hand who works in conjunction with the laboratory	2.58	5.24
(b) Reagents:		
Cresylic acid, 0.77 pound	1.77	3.59
Gas oil, 0.37 pound	.23	.47
(c) Repairs, renewals, stores	.71	1.44
Total operating cost	5.29	10.74

The royalty charged by the Mineral Separation Co. amounted to $3\frac{1}{2}$ pence or 7.1 cents per ton. Forty-five horsepower was required to drive the flotation machine; this was higher than would be necessary if all the feed under the 1/10-inch size were treated. Only about half the quantity of cresylic acid given would be needed at this size. The filter required 45 hp. and the concentrating table, screens, etc., 4 hp. The water required for the whole plant was 200 gallons a minute.

The coke manufactured was an excellent blast-furnace fuel and commanded a premium on the market, besides being salable when the demand is slack. The average analysis showed 91.8 per cent of fixed carbon and 7.7 per cent of ash. The coke was harder than that formerly produced, and less breeze was made. Koppers standard regenerative-type ovens were used. The by-product yield of ammonium sulphate and tar increased while the production of benzol decreased.

11 Exchange at par value.

Another interesting British plant that has operated successfully with a Baum washer has been described by Guider.[12] In this plant, operated by the Clifton & Kersley Coal Co. (Ltd.), a dry 2½-inch slack was screened on a ½-inch screen, the oversize being treated in a separate jig washer from the undersize material. The Baum washer treating the minus ½-inch coal, although removing most of the refuse material, permitted the finely suspended dirt to go over with the washed coal. The dirty suspension, as well as the fine coal, was removed from the minus ½-inch washed coal on a 0.5-mm., wedge-wire, fixed screen, set at an angle of 30° from the horizontal. The 2½ to ½ inch washed coal from the other washer was dewatered and sized in a trommel screen. The sludge that accumulated from both the coarse and the fine dewatering screens went to a sump, from which point it was pumped to a settling tank. The sludge from this settling tank, containing 25.6 per cent solids, was the feed for the flotation machine. A sizing test and float-and-sink analysis of the sludge is shown in Table 5.

The flotation machine, a 6-cell Minerals Separation type, had a capacity of 25 tons per hour. The froth produced by the machine was scraped off by paddle wheels and fell into a trough that delivered it directly to the minus ½-inch jig-washed coal contained in a draining-band bucket conveyor. The coal was then ready for either storage or shipment.

Table 6 shows a size and a float-and-sink analysis of the froth and tailing products.

The material over 1/10 inch was taken out of the tailing by means of a wedge-wire screen and returned to the fine-coal jig. The froth product, while considerably improved in quality, still contained some fine clay, probably due to the fact that a considerable amount of suspended clay comes over with the wet coal from the flotation machine.

Table 5.- Screen and specific-gravity analyses of the feed at a British flotation plant, according to Guider

Size, inches	Weight, per cent	Float in 1.35 specific gravity, per cent	1.35 to 1.60 specific gravity, per cent	Sink in 1.60 specific gravity, per cent
Plus 1/4	1.8	1.6	0.2	-
1/4 to 1/10 .	8.2	6.6	1.2	0.4
1/10 to 1/20.	8.7	6.7	1.3	.7
Minus 1/20 ..	81.3	46.3	18.7	16.3
Total ..	100.0	61.2	21.2	17.4

12 Guider, W., Froth Flotation Applied to a Baum Washer: Jour. Soc. Chem. Ind., Trans., vol. 46, 1927, pp. 238-242.

Table 6.- Screen and specific-gravity analyses of the flotation products at a British plant, according to Guider

FROTH

Size, inches	Weight, per cent	Float in 1.35 specific gravity, per cent	1.35 to 1.60 specific gravity, per cent	Sink in 1.60 specific gravity, per cent
Plus 1/20 ..	5.0	4.5	0.5	-
1/20 to 1/60	38.0	33.5	4.0	0.5
Minus 1/60..	57.0	37.0	15.0	5.0
Total..	100.0	75.0	19.8	5.5
TAILING				
Plus 1/10 ..	1.6	1.0	0.3	0.3
1/10 to 1/20	4.2	1.4	1.4	1.4
Minus 1/20..	94.2	3.4	7.9	82.9
Total..	100.0	5.8	9.6	84.6

The reagent utilized at this plant was a rather crude quality of creosote oil. After trying several types of reagents, it was found that a cheap grade of creosote oil (1.7 pound per ton) gave better and more economical results than other more expensive reagents such as cresylic acid. The cost of operating the flotation plant per ton of coal was as follows:

	Pence	Cents[12a]
Interest on capital (5 per cent)	5.60	11.38
Depreciation (7 per cent)	7.84	15.90
Royalty	4.00	8.12
Power	3.84	7.80
Wages (one man)	1.85	3.76
Reagents	1.50	3.04
Total	24.63	50.00

The increased cost per ton of total slack (through ½ inch) was stated to be 6.1 pence or 12.4 cents as result of the installation of the flotation plant; but the ash content of the washed slack was reduced from 10.0 per cent to 6.0 per cent by the change. The high ash content of the jig product was due to the fact that the raw coal contained a large amount of shale that easily disintegrated to a finely divided clay that went over with the jig-washed coal because the Baum washer is unable to separate such fine dirt from fine coal. Froth-flotation removed most of the clay suspension without a large loss of fine coal. The loss of coal in the flotation tailing amounted to only 0.1 per cent of the total slack (through ½ inch) fed to the washer.

12a Exchange at par value.

Guider considers the major disadvantage of the flotation system to be the fact that the coal froth retains such a large amount of moisture. Water remains with the fine coal so tenaciously that even after mixing with the coarser through ½-inch coal and draining for a period of 30 minutes, the mixture has a moisture content of 18 per cent.

Froth-Flotation of Indian Coals

Randall[13] has investigated the possibility of cleaning Indian coals by the froth-flotation process. A large proportion of the mineral matter contained with Indian coals is very intimately mixed with the coal substance and for this reason cleaning is difficult by ordinary washing methods. Randall states that "tests at various meshes show that to set free the constituent bands of Indian coals it is necessary to crush the coals to about 1/20 inch." For the reason just cited, all of the flotation tests were made with coal crushed to pass a 1/20-inch screen.

Typical results obtained by Randall in a laboratory flotation machine on face samples considered to be representative of run-of-mine coal are shown in Table 7.

Randall's work also showed the ash contents at higher yields of coal. At yields of 80 per cent, the froth products increase 3 or 4 per cent over the values shown for 60 per cent yields; the refuse products likewise show an increase of from 10 to 20 per cent in ash contents. Nevertheless, the ash contents of the refuse products remain rather low even at comparatively high yields of froth concentrates, due no doubt to the bony nature of the coals that were being treated.

Table 7.- Froth flotation tests of Indian coking coals, according to Randall

Description of coal	Raw coal ash, per cent	Washed coal Weight, per cent	Washed coal Ash, per cent	Refuse Weight, per cent	Refuse Ash, per cent
Jharia No. 18 seam	16.3	60	9.4	40	26.7
Jharia No. 13 seam	18.0	60	10.5	40	29.3
Jharia No. 16 seam	22.4	60	11.8	40	38.3
Bokaro, Kargali seam	22.3	60	15.8	40	32.1
Bokaro, Kargali seam 1/	20.6	60	10.5	40	35.8
Barakar, Ramnagar seam	13.6	60	7.6	40	22.6

1/ One-half-inch slack.

The bright coal was more friable and had a much lower ash content than the dull coal. The slack formed in mining was therefore of a higher grade than the run-of-mine coal. For this reason Randall advised the use of preferential crushing for the run-of-mine coal. The minus ½-inch slack should then be used as the raw material for making coke while the plus ½-inch material should be used for steam firing.

13 Randall, W., Froth Flotation of Indian Coals: Rec. Geol. Survey India, vol. 56, part 3, 1925, pp. 220-249.

In France and Belgium

About 180,000 tons of coal were cleaned by froth flotation in France in 1926. Since that time many other plants have been built both in France and in other parts of Europe, and the yearly production of floated coal therefore has increased accordingly.

At the Noeux colliery in France a plant of 1,000 metric tons (1 metric ton is 2,204.6 pounds) capacity per 24-hour day has been built for treatment of a heap of washery refuse.[14] The material was dug by shovel and sent to the flotation plant by aerial tram. Here it was passed first through a 20-mm. vibrating screen of the Hum-mer type (capacity 60 tons per hour). The oversize, which contained more than 75 per cent of ash, was discarded. The remainder, which already contained 8 per cent of moisture, was pulverized wet in a Jeffrey swing-hammer pulverizer (capacity 500 tons per day; another of 700 tons capacity was being installed). It was then screened wet through a 2-mm. sieve (which will be replaced by a 2.5-mm. sieve), the larger material (over 74 per cent of ash) being again rejected. The fine material passes with five times its weight of water to four Minerals Separation, standard 5-compartment, froth-flotation washers, each of 250 tons daily capacity (two in use when the article was written). The froth was dewatered from 60 to 15 per cent of moisture in two Oliver continuous, vacuum filters, each of 20 square meters surface. The waste waters were treated in Dorr thickeners; the dirt was discarded in the form of a thick mud with 50 per cent moisture. The feed to the flotation cells contained 61.16 per cent ash. The yield of recovered coal was 23.3 per cent, and its ash content was 20.28 per cent. The process appears to be successful, but the exact cost per ton treated has not been stated.

Berthelot[15] gives the flotation results for two plants installed by the Minerals Separation Co. in Belgium. Tables 8 and 9 show the results at these two plants which are located at Charleroi and at Bonnier.

Berthelot gives the cost figure for treating a ton of fine raw coal as about 3 francs or 58 cents. He says that the installation should pay for itself within a year.

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14 Chataignon, M., La flotation en son application au terril de Noeux: Rev. Ind. Min., vol. 4, 1924, p. 361; abstract in Fuel and Science and Practice, vol. 3, 1924, p. 453.
Pirlot, F., Application of Flotation to the Treatment of Coal Schists of the Slack of Noeux: Rev. universelle mines, vol. 7, part 4, 1924, pp. 97-107.

15 See footnote 4.

Table 8.- Flotation results at Charleroi, according to Berthelot 1/

	Per cent
Ash content of the raw slimes	30
Ash content of the washed slimes	8
Ash content of the refuse	70
Yield of washed slimes	65
Yield of refuse	35
Moisture after dewatering	16 to 18

1/ Reagents: Mixture of creosote and wood-tar oils, 800 to 900 grams (1.8 to 2 lbs.) per ton of raw coal.

Table 9.- Flotation results at the Bonnier coal mine, according to Berthelot 1/

	Per cent
Ash content of 0 to 2½ mm. of raw coal	30
Ash content of 0 to 2½ mm. of washed coal	7.5
Ash content of middling product	20
Ash content of middling and concentrate together	9
Ash content of refuse	76
Yield of combined middling and concentrate	68
Yield of refuse	32

1/ Reagents: 500 grams (1.1 lbs.) gas oil and 100 to 150 grams (0.22 to 0.33 lbs.) petroleum oil per ton of raw coal.

Dutch State Mines

The froth-flotation process invented by Kleinbentinck,[16] the chief engineer of the Dutch State mines, is used at many of the Dutch mines. At the Emma mine, six Kleinbentinck units having a total capacity of from 12 to 15 tons of sludge per hour, treated a raw sludge with 25 per cent ash. From the first unit, a froth containing 8.5 per cent ash was obtained, and from the second, a froth containing 7.0 per cent ash. The combined remainder from the two units had an ash content of from 36 to 40 per cent. Upon re-treating this remainder, a product containing 10 to 12 per cent ash was obtained, leaving a refuse with an ash content of 57 per cent.

16 Chapman, W. R., and Mott, R. A., The Cleaning of Coal: Chapman & Hall (Ltd.), London, 1928, p. 431.
Stavorinus, D., The Purification of Coal by Foam Flotation: Het Gas, vol. 45, 1925, pp. 266-270.

According to the inventor, the Kleinbentinck process requires 2.7 hp. per ton of raw coal treated. The reagent used by the Dutch State mines was a mixture of one part of napthalene oil and two parts of anthracene oil, both oils being by-products of tar distillation. One and three-quarters pounds of reagent were required per ton of sludge.

The Kleinbentinck process, besides being used by the Dutch State mines, has found application in other parts of Europe. At the Aniche mine, France, three sets of Kleinbentinck units, each set having a capacity of 15 tons of sludge per hour, were installed. The sets were so arranged that the tailing from the first set was rewashed in the second set and the tailing from the second set was rewashed in the third set. In this manner three froth products were obtained with ash contents of 7.5, 11.0, and 18 per cent. Exclusive of the power required for pumping, 4 hp. per ton was required for the throughput of 15 tons per hour. Reagent consumption was 1.6 to 1.8 pounds per ton of sludge treated. In this case a tar-oil distillate boiling between 230° and 270° C. was used as the reagent.

In Germany

Schäfer[17] describes the introduction of the froth-flotation process into Germany for the cleaning of bituminous coal sludges as a utilitarian move. Before the introduction of the flotation process as much of the high-ash and high-sulphur sludges as could possible be used were added directly to the coking coal. The remainder that could not be utilized satisfactorily was sent to waste. The froth-flotation process made it possible to use a much greater percentage of the sludges without materially increasing the ash and sulphur contents of the coke. As an example, Schäfer cites an improvement in the cleaning of a coking coal from the Waldenburg district. By the usual washing process, a 56 per cent yield of material 0.5 to 10 millimeters in size with an ash content of 5.5 per cent was obtained. After the flotation process was introduced as an aid to the washer, the yield was increased to 68.2 per cent and the ash content remained the same as before. In addition, it was shown that by saving some of the slightly higher ash coal, a yield of 78 per cent could be obtained with a total ash content of 6.5 per cent. Another similar example was cited for a high-volatile coal from the Rhine-Westphalian district.

The first commercial plant using the froth-flotation process for cleaning coal in Germany was built at the Mont Cenis coal mine. Wüster[18] has described the Gröndal and France flotation process which is being used at the Mont Cenis

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17 Schäfer, O., Treatment of Coal Slimes by Froth-Flotation Process: Stahl u. Eisen, vol. 45, 1925, pp. 1-7 and 44-51.
Schranz, H., Coal and Ore-Flotation Methods: Krupp Monatsh., vol. 6, 1925, pp. 57-64.

18 Wüster, R., Die Schwimmaufbereitung von Kohle nach dem Verfahren von Gröndall und Franz auf der Zeche Mont Cenis: Glückauf, Bd. 60, 1924, pp. 19-23; Chem. Abs., vol. 18, 1924, p. 1188.

mine for the treatment of coal sludge. Two grades of coal were produced, one for coking and the other for steaming. Five tons of sludge were produced for each 125 to 150 tons of coal washed. By using 470 to 480 grams (about 1 pound) of oil per ton, the coking coal recovered contained about 7.5 per cent of ash, the steam coal about 14.5 of ash, and the refuse 75.8 per cent of ash. The yield was 2.27 tons per 5-ton unit, and the ratio of coking coal to steam coal was approximately 2 to 1. The sulphur content decreased from 2.6 per cent in the sludge feed to 0.77 per cent in the coking coal; the refuse contained 4.3 per cent of sulphur.

The general results obtained at five German flotation plants, as recorded by Schäfer, are shown in Table 10. The first three plants had 10-cell, Minerals Separation type flotation machines, while the fourth plant had an 8-cell unit of the same type. The plant at the Mont Cenis mine has already been described as of the Gröndal design.

Table 11 shows the cost figures for the first four plants. The cost figures for the Mont Cenis flotation plant are not given.

Table 10.- Results at German coal flotation plants, recorded by Schäfer

Description	Ash in raw sludge, per cent	Ash in concentrate, per cent	Ash in refuse, per cent	Yield of concentrate, per cent	Capacity, tons per hour
Zwickau, Saxony	35-40	8-9	70-75	50-55	12
Mölke, Silesia 1/	20-23	8-9	1/30-35	45-50	35
Gelsenkirchen, Westphalia	9-10	5-6	75-80	92-94	15
Altenwald, Saar	22-25	7-8	65-75	70-72	10
Mont Cenis, Westphalia	25-30	7.5	75	2/About 50	5

1/ In this case the separation was intentionally made to yield a concentrate with an 8 to 9 per cent ash content and a middling product with 30 to 35 per cent ash content.

2/ In addition a 24 per cent yield of a middling product containing 14.5 per cent ash was obtained.

Table 11.- Operating costs of German flotation plants per ton of raw coal treated, recorded by Schäfer 1/

Description	Power		Reagents		Wages		Repairs		Total	
	Marks	Cents	Marks	Cents	Marks	Cents	Marks	Cents	Marks	Cents
Zwickau, Saxony	0.13	3.1	0.10	2.4	0.05	1.2	0.06	1.4	0.34	8.1
Mölke, Silesia.	.03	.7	.10	2.4	.02	.5	.02	.5	.17	4.1
Gelsenkirchen, Westphalia ...	.08	1.9	.10	2.4	.04	.9	.05	1.2	.27	6.4
Altenwald, Saar	.13	3.1	.10	2.4	.06	1.4	.07	1.7	.36	8.6

1/ Exchange at par value.

Schäfer also gives the cost of removing the excess water from the concentrate containing 50 to 70 per cent moisture by the use of a pressure-type filter press that removes the water down to 18 per cent. The cost is about 0.8 mark or 19 cents per ton. In this case the cost of removing the excess water from the fine coal is more than twice that for the removal of the dirt alone.

Lucke[19] describes the introduction of the froth flotation system for cleaning coal as a distinct move toward economy. He states that at a bituminous coal washery in the Ruhr district an increase of 3 per cent in the total production of washed coal was brought about by the use of flotation; at gas-coal washeries 10 per cent increase, at Saar and Wurm coal washeries 12 to 15 per cent increase, and at Saxon and Silesia coal washeries a total increase of 20 per cent or more in yield was recorded. In eastern Germany in the Saar and Wurm districts the flotation of coal sludges has been unusually successful, and for this reason the process has a very firm foothold in these districts.

Lucke describes experimental work conducted by the Central European Flotation Preparation A. G. at the research plant of the Humboldt Machine Construction Co. The Humboldt Co., located at Köln-Kalk, constructs flotation machines under the patents of Minerals Separation (Ltd.). The experimental work was carried out with a representative sample of a fine high-volatile coal from a coal washery operating at a mine in the Ruhr district. The washery had a daily capacity of 1,300 tons. Table 12 shows screen and ash analyses of the raw coal and the products of the washery which did not use flotation.

The results indicate that the washing operation is rather effective down to the 0.5-mm. size, but below that size no improvement is shown. For this reason the raw fine coal was screened on a 0.5-mm. screen, and only the material which passed the screen was used for flotation testing. Table 13 shows flotation results on the minus 0.5-mm. coal.

Table 12.- Sieve and ash analyses of raw and washed fine coal from the Ruhr district

Grain size, millimeters	Raw fine coal		Washed fine coal		Fine refuse	
	Weight, per cent	Ash, per cent	Weight, per cent	Ash, per cent	Weight, per cent	Ash, per cent
2 to 8	78.0	12.1	54.8	4.0	44.0	69.3
1 to 2			21.2	6.2	34.7	66.7
0.5 to 1			8.9	9.7	7.4	69.2
0.3 to 0.5	4.4	14.7	3.1	16.3	3.6	64.6
0 to 0.3	17.6	22.6	12.0	26.1	10.3	60.5
0 to 8	100.0	14.0	100.0	8.0	100.0	67.4

19 Lucke, M., Higher Yields of Fine Coal by the Use of Flotation: Ztschr. Verein deutcher Ing., vol. 73, 1929, pp. 1345-1349.
Pieters, H. A. J., The Flotation of Coal Sludges: Brenstoff Chem., vol. 12, 1931, pp. 325-327.

Table 13.- Flotation results on minus 0.5 mm. fine coal from the Ruhr district, according to Lucke

Product	Weight, per cent	Ash, per cent
Minus 0.5 mm. raw fine coal	100.0	21.0
Flotation concentrate	77.3	6.8
Refuse	22.7	69.3

These data show that the flotation process effectively cleans the material that is relatively unimproved by the washing process used at this plant. For the washing plant in question it was calculated that the total yield of fine coal would be increased by 11.5 per cent, or, on a daily tonnage basis, about 130 tons of coal or more would be saved.

An economical study of the losses that were occurring at the washing plant was also made. By estimating the value of the untreated sludge at 3 marks or 68 cents per ton, and the value of the coal that could be saved by flotation at 14.5 marks or $3.35 per ton, the value of the two products for a year would amount to 117,000 marks or about $27,870 for the untreated sludge and 652,500 marks or about $155,400 for the floated coal. The operating cost of the flotation process for a year, judging from the practice in the district, was estimated at 23,100 marks or about $5,500 (0.51 marks or 12 cents per ton of concentrate produced). The yearly loss that was estimated to be occurring at this plant was therefore 512,400 marks or about $122,000. Lucke states that in Germany the average flotation plant, including the cost for dewatering apparatus, will pay for itself in 8 to 12 months of operation.

This author describes a horizontal-type centrifugal dryer, which he calls a rapid dewaterer, as giving very satisfactory results with coal sludges. The dewaterer, simply described, consists of a horizontal, trommel-type, revolving screen into which the coal is delivered. Centrifugal force causes the water to pass through the screen. A spiral scraper within the dewatering screen keeps the fine coal continually moving through the apparatus. The water passing through the dewaterer is sent back to the flotation circuit for reuse. The dewaterer works especially well when the floated coal is first mixed with a larger size of fine washed coal prior to the dewatering. A mixture of 10 per cent of floated coal and 90 per cent of fine coal was reduced in moisture content in this apparatus to 8.8 per cent. This product was then dry enough for immediate coking.

As a general rule, the addition of floated coal to fine washed coal prior to coking increases the yield and the strength of the high-temperature coke. Table 14 gives results showing the effect of floated coal upon the strength of coke. The method of testing was not given.

The normal amount of floated coal that was available for addition to the fine washed coal at the plant where the tests, shown in Table 14, were made was 12 to 15 per cent.

Table 14.- Effect of floated coal upon the strength of coke, recorded by Lucke

Description	Strength, kilograms, per square centimeter
Coke from washed fine coal	124
Coke from washed fine coal plus 5 per cent floated coal ..	138
Coke from washed fine coal plus 10 per cent floated coal ..	150
Coke from washed fine coal plus 15 per cent floated coal ..	163
Coke from floated coal	220

Another German author states that fine coal (0 to 0.5 mm. size) can be cleaned of ash and sulphur very satisfactorily by flotation.[20] The product, however, will contain about 25 per cent water, after filtration, and most of the fusain present in the original coal. Experiments are described which show the feasibility of mixing this product with coarser coal intended for coke manufacture. Coals running 24 to 32 per cent volatile matter are used in the tests. Volatile matter in floated fines ran 2 to 4 per cent lower than that of the coarse coal, mainly because of concentration of fusain in the former. This decreased the coking power in some cases but improved it in others. Bräuer thinks the addition of fines tends to favor cementation of the whole carbonizing mass into a dense hard coke. In one case the addition of 20 per cent of 0 to 0.3 mm. floated coal to the charge increased the stability from 78.3 to 80.5 per cent and decreased the porosity from 41 to 39.4 per cent. Practical methods for mixing the fine coal with the rest of the charge are discussed. The resultant mixture should not contain over 9.5 per cent water, and mixing should be thorough, since segregation of the fines will often cause weak spots in the resultant coke. Good practice is to add the fines from the filter (containing 20 to 25 per cent water) in a thin stream to the rest of the coal while it is being forwarded to the charging wagons on a belt conveyor.

In the United States and Canada

In the United States the use of the froth-flotation process for cleaning coal has not spread as rapidly as it has in Europe. In fact only a few flotation plants have been installed for this purpose. A small washery constructed in 1926 at Snoqualmie, Wash., treated washery sludge by froth flotation with satisfactory results; but this plant was operated only at irregular intervals

20 Bräuer, O., Utilization of Floated Coal in Coke Ovens: Glückauf, vol. 67, 1931, pp. 657-662. Abstracted by J. D. Davis, Chem. Abs., vol. 25, 1931, p. 5749.

because of other unfavorable conditions. Two washeries on Vancouver Island, British Columbia, have used the process in the past, one at Nanaimo[21] for treating slack, sludge, and picking-table refuse, and the other at Cassidy[22] for recovering coal from a high-ash washery sludge.

An experimental laboratory study of the problems encountered in the froth flotation of coal, particularly with reference to the finding of reagents suitable for the depression of pyrite is in progress at the U. S. Bureau of Mines, Northwest Experiment Station at Seattle, Wash., in cooperation with the College of Mines, University of Washington. The compilation of the literature on froth flotation reported in this Information Circular has been used as a basis for the experimental study. These studies have not been completed and have not yet appeared in published form, but the most effective and economical depressants of pyrite found so far in the work are ferrous and ferric sulphates when used in the hydrogen ion concentration range between 4.5 to 6.9. After ferric sulphate has once come into contact with pyrite in a coal pulp in this range it affords depressive action even if the pulp is then carried to the alkaline side of neutrality by the addition of calcium hydroxide or other alkaline reagents.

In general, depressing reagents for pyrite are greatly hampered in their action in the flotation of coal by two factors. These factors are, first, the mechanical entrapment of pyrite particles by the enormous number of coal particles constantly going upward into the froth, and, second, the amount of fine-sized particles of pyrite that remain suspended in the water that accompanies the coal froths. Such detrimental action can be minimized by dilution of the pulp prior to flotation or by the recleaning of the coal froths.

The Pittsburgh Coal Co., Pittsburgh, Pa., has made intensive laboratory and plant investigations of the suitability of the froth-flotation process for cleaning coal sludge. Their results have not been published, but through the courtesy of J. B. Morrow, preparation manager, the following summary of their experimental work prepared by Zimmerman, Younkins, and Crawford of the company's preparation department technical staff, has been made available:

This outline of the experimental work on the froth flotation of coal conducted by the Pittsburgh Coal Co. summarizes the laboratory testing of certain coals and the investigation of various reagents as to their applicability in coal flotation.

Laboratory procedures were governed according to the usual accepted methods, paying close attention to pulp density, agitation, conditioning, reagents and step oiling, water supply, etc. In order to obtain check results, tests were run according to a standard set of procedures.

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21 Coal Age, Flotation Plant Will Treat Slack, Sludge, and Picking-Table Refuse: Vol. 21, 1922, pp. 365-366.

22 Peterson, P. E., Cleaning Vancouver Island Coals by Froth Flotation Process: Coal Age, vol. 21, 1921, pp. 243-244.

Although many coals and cleaning plant products were tested by flotation, the most important material was the Dorr thickener underflow products from the company's three cleaning plants, Champion 1, Champion 4, and Champion 5. This product was uncleaned by the regular coal-washing system used in cleaning the coarser sizes of coal and was approximately of the right size for flotation without any further sizing. Thickener underflow varies in the percentages of sizes present and in ash and sulphur contents according to the plant. Champion 1 has the highest ash and sulphur content and percentage of plus 48-mesh coal. Champion 4 and Champion 5 have lower-ash thickener underflows and comparatively little sulphur. Fifty to sixty per cent of the sulphur is organic and sulphate sulphur and as such can not be removed by flotation. The rest is in the form of pyrite and marcasite.

Particles in thickener underflows larger than 48 mesh were found to interfere with the flotation of the finer sizes, particularly when the finer sizes were extremely high in ash and the plus 48-mesh material low in ash, as is the case at all three plants. Much of the minus 200-mesh material is in the form of slimes. There is also a high percentage of colloids in the thickener-overflow water used in flotation. Both slimes and colloids interfered with flotation of thickener underflow. The most effective methods for dealing with these impurities were by means of greater pulp dilution, recleaning of concentrates, and the use of acid circuits. Sodium silicate showed a tendency to disperse the slimes but affected recovery and caused pyrite to float.

Acid circuits were found to give better-grade concentrates, although maximum recovery was reached at a slightly alkaline (7.0 to 7.5 pH) condition. At higher pH concentrations the recovery and grade dropped appreciably, particularly after 8.5.

The amount of soluble salts and alkalinity of the water supply varies at each of the three plants. Champion 1 water has the lowest amount of soluble salts and only slight alkalinity. The water at Champion 4 has an alkalinity of 160 parts per million and 1,596 parts per million of soluble salts. Champion 5 is the only plant whose water supply is on the acid side, with 35 parts per million of free acid and a soluble salt content midway between that of the other two plants.

None of the plants has water which inhibits flotation. The excessive amount of soluble salts at Champion 4 caused a slight retardation to flotation of coal but the effect was not as nearly marked as is evident with excessive slimes and colloids.

Sulphur reduction was important only at the Champion 1 plant. Here a thickener underflow with 3 per cent sulphur made it essential to remove as much of sulphur as possible. Fifty per cent of the total sulphur content of the underflow product at Champion 1 is in the form of organic and sulphate sulphur. The remainder is pyrite and marcasite. The depression of pyrite-marcasite increases with increase of pH. Depressing reagents recommended for

pyrite in coal flotation are lime and iron sulphates. Laminated particles of coal and pyrite are difficult to depress. Oxidized pyrite, however, aids in the depression of pyrite.

In the cleaning of coal by froth flotation, mechanical entrapment of impurities, including pyrite, in the froth was found to be one of the most important factors in interfering with an efficient separation since 80 to 90 per cent of the feed is floated. In order to compensate for this at least one recleaning of the concentrates is recommended.

Cresylic acid was found to be the best frothing agent for our purpose although pine oil and flotation aldol gave good grades of concentrates except with respect to sulphur content. An important factor in choosing a frothing agent was the need for a brittle froth. Pennsylvania crude oil, kerosene, solar oil, and water-gas tar were found to be satisfactory collectors in the flotation of coal.

According to Parmley,[23] the clean coal obtained by froth-flotation treatment of thickener underflow at the plants of the Pittsburgh Coal Co. is dewatered to about 21 per cent moisture by Oliver, Dorr, and Laughlin type filters. The filter cake is further dried to 3 per cent moisture by means of rotary heat dryers after mixture with the 0 to 3/8 inch size of washed coal discharged from Carpenter centrifugal dryers. The performance of the heat dryers and the size composition of the components of the feed, including the flotation coal, is shown in Table 15.

The cost of heat-drying the fine coal (0 to 3/8 inch) at this plant, which amounts to about 14 per cent of the washer feed, is given by Parmley as 7 cents per ton of dry coal. This figure represents operating costs and supplies only and does not include depreciation and overhead charges. The power and fuel required in the drying plant are given as 1.60 kw.-hr. and 15 pounds, respectively, per ton of coal dried.

23 Parmley, S. M., Heat Drying of Washed Coal: Trans. Am. Inst. Min. and Met. Eng., Coal Division, vol. 94, 1931, pp. 336-350.

Table 15.- Coal fed to three heat dryers, according to Parmley

	Centrifugal dryer product, 0 by 3/8 inch	Filter product	Composite feed to heat dryer	Discharge from three cyclones
Tons per hour	100 to 120	20 to 25	120 to 145	3
Moisture in feed, per cent	6.5 to 8.5	21	7.5 to 9.5	-
Moisture in discharge, per cent	--	--	3.0	1.5
Screen test, per cent weight				
On 3/8 inch round	0.0	0.0	0.0	0.0
14 mesh 1/	72.0	0.0	62.0	20.0
48 mesh	20.5	33.0	22.0	48.0
100 mesh	3.5	31.0	7.5	22.0
200 mesh	1.5	14.0	3.5	6.0
-200 mesh	2.5	22.0	5.0	4.0
	100.0	100.0	100.0	100.0

1/ Tyler standard sieves.

Elmore Vacuum Process

Chapman[24] states that one of the outstanding recent developments in the wet cleaning of coal is the introduction of the Elmore vacuum-flotation process, formerly used in ore flotation. The ordinary type of flotation machine requires a large amount of power for the violent agitation needed. The rather stable froths produced by these machines are difficult to break down and to dewater. The Elmore process, according to this author, eliminates these major disadvantages of the mechanical process by operating under a vacuum without agitation.

In the Elmore process the raw coal (under 1/8 inch) is mixed with about six times its weight of water together with a small amount of suitable reagents. The feed is then delivered into the frothing chamber through a conical barometric leg. The feed pipe gradually increases in diameter toward the frothing chamber to allow for the free expansion of the pulp caused by the low pressure in the frothing chamber. The pressure in the main body of the apparatus is 24 to 27 inches of mercury below atmospheric. The reduced pressure causes the formation of air bubbles from three general sources: First, air dissolved in the water; second, air released from the structure of the coal particle; third, air attached to the oiled-coal particles during the mixing. The general result is a floating of the coal particles upward into an overflow pipe, where they are removed, while at the same time the refuse particles are

24 Chapman, W. R., Recent Progress in Coal-Cleaning Practice in Great Britain: Proc. Third Internat. Conf. Bit. Coal, vol. 2, 1931, pp. 741-758.

left behind and are separately removed by a pipe attached to the bottom of the vacuum chamber. The two outlet pipes (over 30 ft. in height) carry their separate products out of the flotation chamber to atmospheric pressure. The froth from the vacuum process collapses immediately upon leaving the vacuum chamber and by natural drainage the coal below 1/8-inch in size can be reduced to 12 per cent moisture in two hours. This is considered to be about half the normal amount usually retained by minus 1/8-inch coal. The power requirements for the operation of the process are very low.

Chapman does not describe a commercial plant employing the Elmore process but states that the process has been shown, on a commercial scale, to give better results than normal present practice. Table 16 shows experimental results obtained with this process.

Table 16.- Purification of fine coal in experimental vacuum flotation plant, according to Chapman 1/

Description	Ash, per cent		
	Raw coal	Clean coal	Refuse
Scotch slurry	15.4	4.2	58.5
Kent coal	8.8	3.8	56.0

1/ Chapman, W. R., and Mott, R. A., The Cleaning of Coal: Chapman and Hall (Ltd.), London, 1928, p. 436.

Some Recent Comments

As an example of the remarkable results that can be obtained by the froth-flotation process, Chapman[25] is quoted as follows: "At a colliery in the north of England coal is being cleaned by froth flotation and coked, and the cleaning is so effective that the coke is sold under guarantee with not more than 1 per cent of ash."

Berthelot[26] states: "Two large plants, the Randolf Colliery and the Dumbreck Colliery, use flotation coal for making supercoke containing only 1.5 per cent ash, which is used for the manufacture of electrodes."

Chapman further states that the use of the flotation process for cleaning coal has been held back for two main reasons: First, the ordinary type of machine requires a large amount of power for operation; second, coal froths are difficult to dewater. He claims, however, that since the process is the only positive method for cleaning coal sludges, it is again coming into prominence for cleaning coal in Great Britain. He also predicts that the recent development of the Elmore vacuum-flotation process will eliminate most of the previous difficulties encountered with the ordinary type of flotation machine.

25 See footnote 24.
26 Berthelot, C., Modern Methods of Washing Coal, Especially Fine Coal: Proc. Third Internat. Conf. Bit. Coal, vol. 2, 1931, p. 761.

TRENT PROCESS

Experimental Investigation and Principle of the Process

The Trent process, invented by Walter E. Trent, makes use of the selective wetting of coal by oil and of mineral matter by water. The raw fine coal (through 100 mesh or finer) is suspended in water and then mixed with oil amounting to 30 or 40 per cent of the coal. The coal particles form with the oil a pasty agglomerate that sinks to the bottom while most of the refuse material remains suspended in the water. The coal mixture may then be removed. The agglomerate, called an amalgam, retains 8 to 12 per cent moisture.

The United States Bureau of Mines has studied the effectiveness of the Trent process in reducing the ash and sulphur content of coal on a laboratory scale[27] and the results of destructive distillation of the coal-oil mixture produced by this method of treatment.[28] The study of the effectiveness of the process in reducing the ash and sulphur content of coal was conducted by Perrott and Kinney, whose report should be consulted. Table 17 following summarizes their results.

Table 17 shows that in many cases the Trent process causes a very effective separation of dirt from coal. When the amalgams are carbonized at a high temperature, exceedingly large amounts of a high-grade gas are produced and, in many instances, a firm coke product is formed even when noncoking coals are used.

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27 Perrott, G. St. J., and Kinney, S. P., Laboratory Studies of the Trent Process: Rept. of Investigations 2263, Bureau of Mines, July, 1921, 18 pp. Reprinted in Chem. and Met. Eng., vol. 25, 1921, pp. 182-188, and Coal Age, vol. 20, 1921, pp. 132-134, 172-175. See also Haanel, B. F., Trent Process for Purifying Coal High in Ash: Canada Dept. Mines, Summary Rept. 574, 1920, pp. 43-44.

28 Davis, J. D., Place, P. B., and Scott, G. S., Destructive Distillation of Mixtures of Oil and Coal: Rept. of Investigations 2301, Bureau of Mines, December, 1921. Reprinted in Chem. and Met. Eng., vol. 25, 1921, pp. 1131-1136.

Davis, J. D., and Coleman, C. E., Low-Temperature Distillation of Amalgams of Noncoking Coal and Asphaltic Oils: Rept. of Investigations 2312, Bureau of Mines, January, 1922. Reprinted in Chem. and Met. Eng., vol. 26, 1922, p. 173.

Davis, J. D., Distillation Gases Yielded by Trent Amalgams and Ethylene Found Therein as a Source of Alcohol: Rept. of Investigations 2415, Bureau of Mines, November, 1922.

Table 17.- Summary of results of Trent process, according to Perrott and Kinney

Kind of coal and source	Oil used, gallons per ton	Raw coal, per cent		Cleaned coal, per cent			Refuse, per cent			Agitation period, hours
		Ash	Sulphur	Weight	Ash	Sulphur	Weight	Ash	Sulphur	
Anthracite:										
Culm I	65	27.7	1.00	74.0	7.0	0.70	26.0	87.0	1.99	0.5
Culm II	65	31.4	1.63	69.0	6.5	.85	31.0	87.0	3.05	.5
Rhode Island	75	21.7	.85	82.0	6.7	.83	18.0	90.7	.95	2.0
Bituminous:										
Pittsburgh bed	80	12.5	1.27	92.0	6.0	1.34	8.0	88.0	.40	.1
Upper Freeport bed.	80	9.3	2.28	96.5	6.7	2.34	3.5	87.6	.60	2.0
Bone-coal refuse ..	80	21.7	.93	88.0	12.5	.80	12.0	88.7	2.08	1.0
Illinois	80	16.6	5.33	85.0	7.4	5.28	15.0	69.7	2.25	3.0
Indiana	80	9.9	4.38	96.4	6.3	4.27	3.6	86.2	.80	.5
Oklahoma	80	19.5	4.74	69.0	5.7	3.08	31.0	50.5	8.50	2.0
Washington	80	22.6	.49	87.5	13.6	.50	12.5	85.0	.50	.5
Brazil	60	35.6	2.47	66.0	9.4	2.32	34.0	86.0	2.71	4.0
Bituminous refuse:										
New Mexico	60	54.7	.55	45.0	22.9	.86	55.0	80.6	.29	2.0
Tennessee	50	63.5	1.64	31.0	20.6	1.48	69.0	82.7	1.65	2.0
Alabama	80	23.5	1.60	80.5	6.6	1.76	19.5	92.8	.90	1.0
Subbituminous:										
Washington	80	19.3	.48	87.0	10.0	.50	13.0	80.0	.45	3.0
Lignite:										
California 1/	80	35.1	1.77	81.5	25.7	1.56	18.5	75.9	2.30	2.0
Texas 1/	80	33.5	1.44	79.7	18.1	1.42	20.3	94.2	1.25	2.0

1/ Carbonized at 500° C.

Commercial Application

The Trent process may be said to have had only a limited commercial application. According to Davidson,[29] in January, 1925, five plants were manufacturing under Trent patents in the United States and one in France. Because of the high cost of manufacture and because of the increased competition in the market caused by other types of fuels, many of the plants have since closed. The Toledo plant, described by Davidson, in a period of eight months produced some 35,000 tons of bulk amalgam. This plant operated successfully for a period of several years. From 85 cents to $1 screenings the plant manufactured bulk amalgam at a cost of $6 per ton. The freight charges were low for the raw material but exceedingly high for the finished product. Amalgam manufactured at Toledo, Ohio, for $6 per ton, when shipped to Detroit, Mich., could be sold for no less than $11 to $13 per ton. In order to make the product more adaptable for home use many attempts have been made to put the bulk amalgam into the form of clean attractive oil-proof packages, but because of the high cost of manufacture and the failure of the public to adopt the new type of fuel, the process has not been entirely successful from a commerical standpoint.

COMPARISON OF THE FROTH-FLOTATION AND TRENT PROCESSES

A direct comparison of the froth-flotation process with the Trent process when both are applied to the Pacific Northwest coals has been made by the Bureau of Mines at Seattle, by Ralston.[30] Table 18 gives the average results obtained by the two methods when the same coal is treated. All of the coals tested were from the State of Washington and were rather high in inherent ash. Of the five coals tested, two were subbituminous, two bituminous, and one semianthracite. Ralston's work is so well summarized in the conclusions of this paper that they are quoted here in part as follows:

Table 18.- Comparison of work done by Trent and froth-flotation processes, according to Ralston,[1] per cent

Raw-coal ash	Trent process, 300 mesh		Trent process, 65 mesh		Flotation, 65 mesh	
	Concentrate ash	Recovery 2/	Concentrate ash	Recovery 2/	Concentrate ash	Recovery 2/
15.3	9.0	99.5	13.5	99.0	13.8	90.4
20.8	10.6	98.0	13.5	98.0	15.5	80.0
26.5	12.2	98.0	16.3	96.0	18.4	90.0
24.8	15.8	99.5	18.0	98.5	18.3	96.0
13.7	7.7	99.5	10.8	99.0	10.7	97.0

1/ All figures on moisture-free basis.
2/ Recovery of combustible material.

29 Davidson, E. W., Toledo Plant Makes Trent Amalgam Better Known: Coal Age, vol. 27, 1925, pp. 139-142.
Dacy, G. H., Plastic Fuel Can be Made of Low-Grade Coal and Oil and Can be Coked Even if Noncoking Coal is Used: Coal Age, vol. 21, 1922, pp.953-956.
Kneeland, F. H., Plant in Newark Makes Briquets by Trent Process: Coal Age, vol. 26, 1924, p. 715.
30 Ralston, O. C., Comparison of Froth with Trent Process: Coal Age, vol. 22, 1922, pp. 911-914.

1. Froth flotation and the Trent process, when applied to the cleaning of various coals ground to pass a 65-mesh sieve for the purpose of preparing a low-ash fuel, gave very similar results under conditions which were most favorable to the froth-flotation work and somewhat unfavorable to the Trent work.

2. Under these conditions the grades of concentrate made by the two processes were almost identical for all kinds of coals tested, with a slight advantage in favor of the Trent process when heavy oils were used.

3. The extractions of combustible matter are higher in the Trent process, the differences being slight for semianthracite and coking bituminous, but up to 5 or 10 per cent for the lower grades of coal.

4. The Trent process carries the bony coal into the "clean" coal product, whereas it is possible by froth flotation to carry the cleanest coal into a first concentrate and the bony coal can then be taken off in a second concentrate, which can be segregated, if desired. For this reason the froth-flotation process is the more flexible of the two.

SUMMARY

The use of the froth-flotation process has increased the yield and quality of the marketable coal products in all of the plants that have been described. The total plant yields have been increased as much as 3 to 20 per cent. In the cases where the process has been applied to the cleaning of fine coking coal, the resulting coke has been considerably improved with regard to strength, ash content, and, in many instances, sulphur content.

The reagents that have been used at the many different coal flotation installations vary to a considerable extent. Those plants that were located near a by-product coking installation generally made use of coal-tar derivatives such as creosote oil, cresylic acid, tar-oil distillates, and napthalene or anthracene oils as reagents. Other plants employed such products as gas oil, petroleum oil, and wood-tar distillates. The reagent consumption usually amounted to from 1 to 2 pounds per ton of dry raw coal.

The feed to the coal-flotation plants described usually contains 20 to 25 per cent solids. Two products, a clean coal froth and a tailing or refuse product usually are made. The coal froths from the flotation machines contain 50 to 70 per cent moisture. The tailings from the flotation plants usually contain only a small amount of good coal and therefore generally are sent to waste.

Many European flotation-plant operators consider the problem of dewatering flotation concentrates as one of the major disadvantages of the process. At the various plants many methods have been used to dewater the froths. One of the simple methods that has been used has been the pouring of the froth concentrate onto coarser coal contained in a slow-moving drainage conveyor. The moisture content of the mixture was reduced in this way to 18 per cent. The revolving-drum filter has been used to dewater coal-flotation concentrate at a number of plants. According to the literature on the subject, the method is rather expensive but more positive in its action than the simple draining method. The minus 1/10-inch froth concentrate can be dewatered by this method to 15 or 20 per cent moisture. If this material then is mixed with a considerable amount of the through $\frac{1}{2}$-inch coal (1 part of froth to 10 parts of $\frac{1}{2}$-inch coal), the resulting product will contain about 8 to 10 per cent moisture and is suitable for coking purposes. The centrifugal dryer has been used successfully in Germany for dewatering mixtures of coal froths with coarser coal. This process gave results somewhat similar to those obtained with the revolving-drum filter. Other methods of dewatering the froth concentrates have been tried, but, regardless of the method that has been used, it seems to be a well-recognized fact that the cost of removing the excess water from the clean coal concentrates is a large item of expense. In several cases the cost of dewatering the concentrates has equaled or exceeded the cost of the main process itself.

It is difficult to summarize the cost items at the various European plants, due to differences in the manner of presentation of cost data; positive dewatering costs may or may not be included. However, from the information available it may be estimated that the operating cost of a plant which includes cost of power, labor, reagents, repairs, and similar small items, is about 10 to 25 cents per ton. If the cost of depreciation, interest on investment, and royalty charges is included, the cost may amount to as much as 50 or 60 cents per ton of raw coal treated.

PATENTS PERTAINING TO COAL FLOTATION

United States

1,329,493 - Bacon, R. F. Flotation of coal. Feb. 3, 1920.
1,388,868 - Jones, F. B., and Bury, E. Froth flotation. Aug. 30, 1921.
1,418,547 - Edser, E., Sulman, H. L., and Jones, F. B. (Assignors to Minerals Separation North American Corporation.) Treatment of materials containing coal. June 6, 1922.
1,499,872 - Price, F. G. (Assignor to Minerals Separation North American Corporation). Treatment of coal. July 1, 1924.
1,539,746 - Kleinbentinck, J. W. Means for refining coal slimes or the like. May 26, 1925.
1,551,956 - Bates, L. T. Separating ash-forming constituents from coal by flotation. Sept. 1, 1925.
1,578,274 - Eldred, B. E., and Graham, R. N. Flotation separation of coal and ash or similar material. March 30, 1926.
1,595,745 - Truran, W. Concentrating coal-bearing material by repeated froth flotation. Aug. 10, 1926.
1,655,849 - Stenning, W. W. Concentrating coal by froth flotation. Jan. 10, 1928.
1,706,281 - Elmore, F. E. Vacuum system of flotation separation of coal, ores, etc. March 19, 1929.
1,707,429-30 - Chemische Fabrik in Billwärder (formerly Hell et Sthamer A.-G.). Extraction of clay. April 2, 1929.
1,758,756 - Morgan, H. Apparatus for flotation-separation testing of coal mixed with slate. May 13, 1930.
1,787,938 - Eisele, J., Griessbach, R., and Heuch, C. (to I. G. Farbenind A.-G.). Froth flotation of pulps such as those of coals or sandy pyrites. Jan. 6, 1931.

British

159,285 - Edser, E., Sulman, H. L., and Jones, F. B. Recovering coal. Nov. 20, 1919.
186,143 - Price, F. G., and Minerals Separation (Ltd.). Flotation of coal. June 20, 1921.
193,466 - Stenning, W. W., Williams, P. T., Beasley, W. H., and Middleton, A. B. Treating coal. Nov. 18, 1921.
199,753 - Eldred, B. E. Separating materials. Jan. 5, 1922.
208,226 - Williams, P. T., and Minerals Separation (Ltd.), Concentrating coal, etc. Sept. 13, 1922.
215,615 - Withers, J. S., Froth-flotation apparatus for concentrating ores, coal slimes, etc. June 18, 1923.
218,012 - Minerals Separation (Ltd.). Mineral flotation separation. March 29, 1923.
253,618 - Minerals Separation (Ltd.). Froth-flotation machine for coal, ores, etc. 1925.

275,778 - Elmore, F. E. Vacuum-flotation apparatus for concentrating coal, ores, etc. July 16, 1926.
286,456 - Chemische Fabrik in Billwärder (vorm. Hell & Sthamer A.-G.) and Kuhlwein, F. L. Flotation separation of coal from clay or other impurities. March 5, 1927.
289,848 - Schäfer, W., and Erz u. Kohle Flotation G.m.b.H. Flotation of coal, graphite, ores, etc. May 4, 1927.
298,736 - I. G. Farbenind A.-G. Concentrating ores, coal, etc., by flotation. Aug. 19, 1927.
357,733 - Combined Metals Reduction Co. Removing resins from coal by froth flotation. Aug. 22, 1930.

Russian

4,643 - Sleptzov, E. P. Method for sorting coal by flotation. February, 1928.

French

649,270 - Schäfer, W., and Erz u. Kohle Flotation G.m.b.H. Flotation process. Feb. 20, 1928.

German

478,065 - Minerals Separation (Ltd.). Refining coal. July 9, 1920.
493,110 - Minerals Separation (Ltd.). Froth flotation machine for coal, ores, etc. Feb. 5, 1926.

Dutch

524,869 - Vereenigde Kolenmaatschappi Jen (Ter Voortjetting der Steenkolenzaken, Gedreven door de N. V. Furness, Kolenmaatschappij). Flotation apparatus for working up coal slimes. April, 1923.

Department of Commerce

United States Bureau of Mines
Scott Turner, Director

INFORMATION CIRCULAR

MINING LAWS OF CEYLON

BY

E. P. Youngman

I.C. 6715
May, 1933.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING LAWS OF CEYLON[1]

By E. P. Youngman[2]

PREFATORY NOTE

This paper is one of a series of digests of foreign mining legislation and court decisions that is being prepared in advance of a general report relative to the right of American citizens to explore for minerals and to own and operate mines in various foreign countries. This interpretation of the mining laws of Ceylon has been prepared from the best available information in Washington and is released subject to amplification and correction, if necessary, by United States foreign-service officers.

INTRODUCTION

Ceylon has no basic mining law; the general rules governing the granting of mineral concessions and affecting legal titles of concessionaires are laid down in Ceylon Government ordinances or are found in the authorized forms for licenses and leases. The material available in the preparation of this digest is as follows:

Ordinance No. 5 of 1890, the gemming ordinance, or an ordinance relating to mines of gold, silver, gems, and precious stones in lands other than Crown property, as amended by No. 10 of 1894, No. 13 of 1905, and No. 18 of 1908.

Ordinance No. 2 of 1896, the mines and machinery protection ordinance, 1896, as amended by No. 28 of 1908, No. 11 of 1914, and No. 25 of 1918.

General order affecting plumbago lands of different qualities, Sept. 9, 1919.

General order affecting prospecting licenses, Sept. 9, 1919.

1/ The Bureau of Mines will welcome reprinting of this article, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular ."

2/ Rare metals and nonmetals division, U. S. Bureau of Mines.

General order affecting mining leases, Sept. 9, 1919.

Form of general prospecting license, Sept. 9, 1919.

Form of general mining lease, Sept. 9, 1919.

Form of plumbago mining lease, Sept. 9, 1919.

Form of preliminary plumbago mining license, Sept. 9, 1919.

Ordinance No. 8 of 1899, the quarries ordinance, as amended by section 240 of No. 11 of 1920.

Ordinance No. 6 of 1890, an ordinance to consolidate the laws relating to Her Majesty's revenue from salt.

The apparent ease with which mineral rights may be procured in Ceylon should not receive undue emphasis, as the mineral resources of the island are not great, and the public domain is not large, being confined to the mountainous and more or less inaccessible parts. Ceylon was well established before it became a British colony.

No provisions exist for governmental financing of private companies engaged in producing or distributing mineral products.

RIGHTS OF FOREIGNERS

No restrictions, legislative or administrative, are placed upon aliens that are not placed upon nationals in the granting of mining rights or concessions; in the operation of mineral properties; in the distribution of mining products; or in the sale of mining rights and properties.[3]

OWNERSHIP OF MINERALS

The lack of a basic mining law has left some questions open to individual interpretation, among them being the questions of the ownership of minerals and the extent to which Government permission must be procured before mining privileges may be enjoyed.

However, the courts have recognized the right of the Crown to a one-tenth share in all metals and minerals of commercial value raised from private lands, in the form of a royalty. In the case of plumbago, this right has been commuted by legislation and is represented by a duty collected on exports (sec. 1, No. 22 of 1877)--at a rate of 25 cents hundredweight by the act of 1877 and at the rate of 3 per cent ad valorem by act No. 21 of 1916.

3/ Leonard, Walter A., Mining Laws and Legal Restrictions Imposed on Foreigners in Securing or Operating Mineral Concessions: Consular Rept. 129632, Colombo, Oct. 25, 1919, Bureau of Mines foreign file 742.

As the matter of ownership has not been clarified by a definite statement, the following discussion by Pereira[4] is interesting:

> It would, perhaps, be difficult to maintain that the Crown has any more extensive rights to precious metals than are reserved by Ordinance No. 5 of 1890. It has been said that possibly the Crown is entitled to all the rights in respect of mines, minerals, and precious stones granted by the States-General to the "Dutch Federation in India" by the placaat referred to by Voet in Book 41, Title 1, of his Commentary, which gave the company exclusive rights to "all minerals and mines of gold, silver, copper, and other metal, and of precious stones, diamonds, rubies, and the like," reserving to the actual finders only one-twentieth of the proceeds, and that only for the first five years of their working; but the placaat, in terms, applies only to the West Indies, and even supposing that a similar monopoly was granted to the Dutch East India Company, it is doubtful that the Crown can claim to have succeeded to a monopoly conceded by legislation to an extinct private company. In view of the above, when land is granted by the Crown without a reservation of minerals, the minerals in the property are vested in the grantee, subject to a royalty on all minerals taken from the land, or the payment of a duty which is the equivalent of a royalty, and when land is declared to be the property of a claimant under the Waste Lands Ordinance the same result follows: In the Kandyan Provinces, it would appear from the Nitinighanduwa and Armour's Grammar that the Crown has a right to all mines, minerals, precious stones, etc.

Ordinance No. 5 of 1890, in its preamble, states that the purpose of the enactment is "to provide for the better protection of the prerogative rights of the Crown with respect to gold, silver (which are not mined in Ceylon), gems, and precious stones that may be found in private lands. The exact significance of the expression "prerogative rights of the Crown" is not clear. Crown grants are often made with the expressed right of the Crown to the minerals in the land.

Closely allied with the question of ownership of minerals is that of when and by whom licenses and leases to prospect and mine are required.

That permission to mine must be obtained from the Government before it shall be lowful for any person to mine for gold, silver, gems, and precious stones in lands other than Crown property is attested by ordinance No. 5 of 1890. The very existence of this law presupposes that such permission is not necessary with respect to other minerals found on private land, unless the Crown has reserved mineral rights in the land grant.

The prospecting and mining license and lease forms available relate to "lands at the disposal of the Crown" or to "Crown lands."

4/ Pereira, James Cecil, The Laws of Ceylon; Law of Things: Pt. 4, 2d ed., Colombo, 1913, pp. 286-288.

However, Van Wagenen[5] says:

> Prospecting and mining for the precious metals and gem stones is forbidden, except to holders of Government licenses. It is not clear whether these must be obtained by searchers for other minerals, but the presumption is that they should be.

A case decision (Saunders v. Marwood, Co. Rep. 12 a; Clegg v. Rowland, L. R. 2 Eq. 160; 17 C.R.C. 723) seems in part to support Van Wagenen's conclusion. It reads:[6]

> A lessee of land without mention of mines may work open mines but may not open new ones.

If the lessee of the land himself may mine underground only with the permission of the Government, it is to be supposed that every other person is under the same restriction.

With respect to forest land, the consent of the Conservator of Forests and the payment of royalty upon the timber are necessary before a license or lease shall be issued. No timber on licensed or leased forest land shall be cut or injured without the permission of the Conservator of Forests.

REGISTRATION

With respect to registration, see the land registration ordinances, No. 5 of 1877 and No. 4 of 1899, and the ordinance relating to the registration of titles to land and of all deeds affecting land in the colony, No. 14 of 1891 (as amended by No. 13 of 1908, No. 29 of 1917, No. 21 of 1918, No. 11 of 1919, and No. 22 of 1921.)[7]

MONOPOLIES

The only real monopoly in mineral resources is that in salt. The Government exercises direct control over mica and monazite sand, which, however, can be mined for export and are not subject to a monopoly law.

By ordinance No. 6 of 1890,[8] an ordinance to consolidate the laws relating to Her Majesty's revenue from salt, it is not lawful for any person to collect or manufacture salt by any processes whatsoever, except on the account of the Government and under a written license of the Government Agent of the Province or the assistant agent of the district in which it is collected or manufactured.

5/ Van Wagenen, Theo. F., Ceylon: International Mining Law, 1918, pp. 247-249.
6/ Pereira, James Cecil, Work cited.
7/ Revised Edition of the Legislative Enactments of Ceylon, vol. 2, Colombo, 1923, pp. 86-124.
8/ Revised Edition of the Legislative Enactments of Ceylon, vol. 2, Colombo, 1923, pp. 57-61.

GOLD, SILVER, GEMS, AND PRECIOUS STONES ON PRIVATE LAND

General

The purpose of ordinance No. 5 of 1890, according to its preamble, is to "provide for the better protection of the prerogative rights of the Crown with respect to all gold, silver, gems, or precious stones that may be found in mines9/ in private lands10/ and for the regulation and inspection of such mines." The ordinance, by section 1 thereof, is given a short title, "The Gemming Ordinance, 1890," probably because no gold and silver are mined in Ceylon.

This ordinance makes it unlawful for any person (including any association or body of persons whether incorporated or not) to open, work, or use11/ any mine without having obtained a license. (Secs. 2 and 3.)

Any question as to whether a mine is a mine to which the ordinance applies shall be referred to the Governor in Executive Council, whose decision shall be final. (Sec. 14.)

A licensee shall produce his license whenever called upon to do so by the Government Agent of the Province or the assistant agent of the district concerned or by any other person having a written authorization. (Sec. 9.)

Mining License

The right to grant or refuse a mining license lies with the Government Agent of the Province in which the land in question is situated. The agent may attach any conditions and require any security he deems expedient. Appeal against the agent's decision may be taken to the Governor in Executive Council within 30 days of the date of the order; the decision of the Governor is final.

A mining license is issued to any person that has established a prima facie right thereto. (Secs. 4 and 6.)

Application.--An application, made to the Government Agent, shall declare in writing: (1) The name and boundaries of the land in which the mine is to be opened; (2) the nature of the right applied for; and (3) the name or names and residence or residences of the applicant and of the manager or the supervisor.

9/ Mine includes one or more mines opened, worked, or used in any land for which a license has been issued under this ordinance. . . and includes all shafts (which include pits), levels, planes, works, machinery, tramways, and sidings both below and above ground. (Sec. 2.)

10/ All land not the property of the Crown, including the bed of every river and stream adjacent to or flowing through the land. (Sec. 2.)

11/ To open, work, or use a mine means and includes the sinking of any shaft or the driving of any level or inclined plane or any act whereby the soil or earth or any rock, stone, or quartz in or under any land is disturbed, removed, carted, carried, washed, sifted, or otherwise dealt with for the purpose of searching for or obtaining gold, etc. (Sec. 2.)

If the person making the declaration shall cease to have any interest in the mine, or if some one other than the person named in the declaration shall be intrusted with the management or superintendence of the mine, the licensee shall make a further written declaration to that effect.

Every declaration shall be signed by the person making it or by his authorized agent and shall be filed in the office of the Government Agent. (Sec. 5.)

Fee.--A mining license is subject to a fee of one rupee,[12] in the form of stamp duty. (Sec. 4.)

Objections.--Any person claiming a title superior to that of the licensee with respect to all or part of the licensed land may apply to a competent court for a restraining injunction. Should the court uphold the superior title of the objector to the license, the court shall revoke the license. (Sec. 7.)

Offenses and penalties.--Any one opening or causing to be opened any mine in any way contrary to the provisions of the ordinance or of the rules made by the Governor in Council or any licensee refusing to produce his license shall be liable to a fine not to exceed Rs. 50, or to rigorous imprisonment not to exceed three months, or both, or to a fine not to exceed Rs. 100 or imprisonment for six months, or both, for subsequent offenses. (Sec. 11.)

When any person is convicted of opening, working, or using a mine without a license or contrary to its provisions or the provisions of the ordinance, all the gold, silver, gems, precious stones, and mining implements found in the possession of the offender shall be liable to confiscation by the convicting magistrate--this in addition to the other prescribed penalties. (Sec. 12.)

The burden of proof that the person prosecuted holds a license shall lie with the accused person. (Sec. 13.)

A prosecution under this ordinance must be entertained within six months of the date of the commission of the offense. (Sec. 15.)

It shall be lawful for the court imposing a fine to award the informer any share not exceeding half the amount of the fine actually realized. (Sec. 16.)

Transfer.--A mining license is not transferable. (Sec. 9.)

Revocation.--The Government Agent may revoke any license in case the licensee fails to fulfill any of the conditions of his license. (Sec. 6.)

12/ The exchange value in 1931 of the rupee (Indian) was 33.6895 cents.

MINERALS OTHER THAN PLUMBAGO ON CROWN LAND

Prospecting License[13]

General

The Government Agent of a Province, at his discretion, may issue one or more licenses for prospecting purposes on any land or lands at the disposal of the Crown.

A license to prospect (which shall be for a specified area and for a specified mineral) shall confer upon the licensee the right to perform only such work as shall be necessary to prove the existence and value of the mineral. The licensee shall not be permitted to remove from the land any minerals other than samples for analysis and determination of quality; the licensee shall be responsible for the safe storage of any minerals won until the Government Agent shall issue orders as to their disposal.

No prospecting license shall be granted until the applicant shall have deposited a sum not to exceed Rs. 1,000 or shall have given other security (to the satisfaction of the Government Agent) for compliance with the terms of the license.

A fee of Rs. 10 is due for each license or renewal thereof--the fee to go to the Headmen's Reward Fund.

Application

Every application for a prospecting license, which shall be made to the Government Agent, shall contain: (1) The name, residence, and profession of the applicant; (2) the situation and other particulars of the land in question; and (3) the name of the mineral to be sought.

Duration and Renewal

A prospecting license shall be given for one year, with the right of renewal for a further term of one year if the agent is satisfied that the licensee was prevented from testing the land through circumstances beyond his control.

Labor Condition

A licensee shall inform the Government Agent of the number of men to be employed. This number, which shall be incorporated in the license, shall not be exceeded except with the approval of the agent.

13/ Prospecting license form, with general order relating thereto, Sept. 9, 1918.

Notices

A one week's notice shall be given by the licensee to the Government Agent of the proposed date for the commencement suspension, recommencement, or final abandonment of the rights under the license.

Disputes

Any dispute between a licensee and the Government Agent shall be decided by the Governor.

Transfer

No licensee shall transfer his license or any right or interest thereunder without the consent of the Government Agent.

Termination

The Government Agent may summarily revoke a license for a breach on the part of the holder of any condition thereof. All or part of the security deposited shall be forfeited, at the discretion of the agent.

A licensee shall within six months of the termination or abandonment of his license (whichever is first) fill up any pits, holes, or excavations made in the land and shall restore the surface to the extent deemed reasonable and possible by the agent.

Mining Lease[14]

General

The Government Agent for the Province concerned shall grant a mining lease to a prospecting licensee with respect to the claim he has registered under his license.

A lease, which stipulates the mineral to be won, confers upon the lessee (subject to the covenants and provisions of the lease, as set forth in the following paragraphs):--

1. Liberty and power for himself, his agents, servants, and workmen, at any time during the term of the lease, to dig and sink such pits and shafts as shall be proper for getting all the specified mineral, to stack and deposit it (when raised) on the land contiguous to such pits and shafts until it can be conveniently removed, to erect any engine or engines for working or getting the mineral, to make all necessary ditches or drains, to make and use for pedestrians, horses,

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14/ Mining lease form, with general order relating thereto, Sept. 9, 1919.

wagons, or other carriages all necessary and convenient roads within the land specified in the lease for carrying off the mineral, and all other privileges necessary, requisite, or appertaining for or to the finding, raising, working, procuring, and carrying away of such mineral.

2. Liberty and power to cut and appropriate to any purpose connected with opening or working the mine or with preserving or removing the mineral obtained therefrom all such timber growing on the land as may be necessary only for such purposes, upon obtaining permission from the Conservator of Forests and upon making payment to the Government Agent of the royalty collectible for the timber.

The lessee shall give security, for the due performance of the conditions of the lease, in such a sum (not to exceed Rs. 1,000) as the Government Agent shall determine.

Application

An application shall contain (a) the name, residence, and profession of the applicant, and (b) a plan, in duplicate, on a scale of four chains to an inch.

Duration and Renewal

The term of a mining lease shall not exceed 15 years, but the lessce shall have the option of renewing it for another 15-year period, on the same conditions. Application for a renewal shall be made at least six months before the expiration of the original lease.

Area

Each block of land leased shall not exceed 10 acres in extent and shall not be less than 1 acre; provided that in the case of alluvial deposits of precious metals or precious stones, the area of the block shall not exceed 100 feet square; provided also that the Government Agent shall not grant a mining lease to the possessor of a prospecting license with respect to the claim he has registered under the license when such claim is within the limits of any forest either already reserved or proposed by the Government to be reserved without the previous approval in writing of the Conservator of Forests. The length of a block of land under a mining lease shall not exceed four times the breadth. An error subsequently discovered in description or measurement shall not entitle the lessee to compensation therefor.

Working Condition

A lessee shall commence mining as soon as the lease has been completed and shall continue operations until the lease has been terminated.

Rent and Royalty

The lessee shall pay in advance the rent specified in the lease, which sum shall be taken into consideration in ascertaining the royalty.

The lessee shall pay a royalty on all the produce of any mine leased at such rate a ton upon the class of mineral procured as may be fixed by the Governor for each year of the term for which the lease is granted: Provided that the rate of the royalty shall not exceed 5 per cent of the estimated value of the mineral at the mine when ready for exportation or smelting and reducing; provided that the royalty shall be payable only with respect to the actual quantity of mineral that is, during the course of the year for which the royalty is fixed, either removed from the leased area or taken to some smelting or reducing works within the area; and provided further that the lessee shall in every case pay at least Rs. 100 per annum per acre or portion of an acre on the leased premises by way of royalty. The lessee shall pay also all the rates, taxes, and assessments that shall be payable with respect to the demised premises.

Should the lessee fail to pay the rent or royalty within three months of the date upon which it is due, the Government Agent may enter the premises and take as security all or any of the mineral raised or the movable property and may hold them until all claims are paid; if any payment remains unpaid for six calendar months, the Government Agent may cancel the lease and take possession of the premises.

Compensation

The lessee shall make and pay reasonable satisfaction and compensation for any injury done by him and shall indemnify the Government against all claims made by a third party because of such an injury; but the lessee shall not make any claim to compensation against the lessor for or on account of alleged expenses or on any other account whatsoever.

Disputes

The Governor's decision shall be final in any dispute between the Government Agent and the lessee with respect to the lease, any matter connected with the mines, their working or nonworking, or payments agreed upon.

Transfer

A lessee shall not assign his lease or transfer any right or interest thereunder or underlet any of the premises included in the lease without the written consent of the Government Agent.

Cancellation

In addition to non-payment of rent and royalty, a breach of any of the conditions of a lease by the lessee or cessation of work for 12 months shall be cause for cancellation of the lease.

The lessee, within six months after the termination or abandonment of a lease (whichever shall be first), shall fill up all pits, holes, or excavations and restore the surface of the land to the extent considered reasonable or possible by the Government Agent.

Rights of the Lessor (Governor, Acting for the Crown)

The lessee shall allow the Government Agent, the Conservator of Forests, or any agent or servant of the lessor to enter the land to inspect the mines and take account of the produce thereof.

Miscellaneous

A lessee shall, at his own expense, define and maintain in good condition all boundaries of the leased land.

A lessee shall (1) provide proper weighing scales, (2) keep account of mine production, number of persons employed, and quantity and source of timber and firewood used, and (3) allow the Government Agent, the Conservator of Forests, or any person appointed in writing by either of them to examine all accounts.

PLUMBAGO[15]

General

The chief differences between the provisions with respect to mining for plumbago and mining for other minerals are: That a prospective miner purchases his lease at public auction (that is, purchases the land with mining rights); that a preliminary mining license is granted until such time as the annual rent shall have been determined and paid, if difficulty is encountered in arriving at the rent payable; that no security is required from the lessee for the due performance of the rights granted; that the lease is limited to one mineral and that the finding of any mineral substance other than plumbago shall be reported to the Government; that the size of the area is determined in each individual case, no maximum being stipulated in the lease; that the duration of the lease likewise is not limited to 15 years but is stipulated in the lease (generally for 15 years, however); and that the rent or/and royalty are not limited to a certain percentage but are determined by a Board of Assessors.

15/ Preliminary mining-license and mining-lease forms, with general order relating thereto, Sept. 9, 1919.

Mining Lease
(Auction of Plumbago Land)

A general order (September 9, 1919) reads as follows:

> If a block of land is likely to contain plumbago of merely ordinary quality and quantity, it should be advertised for sale outright by public auction with mining rights,[16] at an upset price fixed by the best means of estimation at the Revenue Officer's disposal, the price recently obtained for like lots similarly situated being an important factor.
>
> The Revenue Officer may at his discretion allow an applicant for a plumbago lease to have the boundaries of the portion of land applied for clearly demarcated and shown on the plan (in duplicate) that is to be handed to the Revenue Officer for transmission to the Surveyor-General, who will permanently landmark the land as early as possible.
>
> A Board of Assessors, consisting of the Revenue Officer and the Inspector of Mines, shall determine the amount of the annual rent (which should never be less than Rs. 10 an acre). The Revenue Officer may then accept the first annual payment for the leasehold and allow mining operations to begin, pending the execution of a formal lease.

With respect to land reported to contain valuable plumbago (in quantity and quality), a different lease is put on sale--for a term of five years, on the conditions set forth in the lease, one of the provisions being that the lessee may abandon his lease by giving a six months' notice and by paying the rent due and payable.

Preliminary Mining License

A general order of September 9, 1919, reads as follows:

> In the event of the potential value of the plumbago deposits being difficult to ascertain, the Revenue Officer may issue a license for one year, on such terms of rent or royalty as may be determined by the Board of Assessors, and the Revenue Officer may then allow mining operations to begin. Such a license may be renewed for periods of one year at a time until it is possible for the Board of Assessors to determine the annual rent, when a lease shall be issued.

All plumbago mined shall be stacked and stored at some place near the pit or shaft or at a fixed reasonable distance from the land proposed to be leased.

16/ In the Galle district, if land is not known to contain plumbago and is not near such land, it may be put up for sale without mining rights, subject to the provision that if plumbago is subsequently found on such land the purchaser shall pay for the mining rights.

The licensee shall notify the Government Agent of the place of storage; and no plumbago shall be removed until it shall have been valued by the Chief Headman and the stipulated royalty shall have been paid. Should the royalty not be paid within two weeks of its assessment, the Government Agent shall have the power to enter the land and to hold the minerals or movable property as security.

After an assessment is made, plumbago shall be removed only under a permit from the Government Agent; otherwise it shall be liable to seizure and confiscation.

The provisions in a preliminary plumbago-mining license are practically the same as those incorporated in general prospecting-license and mining-lease forms with respect to the following matters: Fees; security for the due performance of rights granted; disputes; reports as to the number of employees, etc.; government inspection; erecting and maintaining boundary marks; transfer; and revocation.

QUARRIES, WITHIN CERTAIN AREAS

The quarries ordinance No. 8 of 1889 (as modified by section 240 of No. 11 of 1920) provides that it shall not be lawful for any person without a license to open, work, or use any new or existing quarry (of rock, stone, cabook, or gravel) within the limits of any town where a municipal council is established or within an "urban area in a district council."

The chairman of a municipal council or the chairman of the district council may refuse or grant a quarry license at his discretion, may attach such conditions and take such security as to him seem expedient, and may revoke any license for a breach of its conditions.

MINING REGULATIONS

Mines Other Than Those of Gold, Silver, and Precious Stones on Private Land 17/

Ordinance No. 2 of 1896 relates to all mines for the purpose of searching for or obtaining minerals of every description (except those of gold, silver, gems, and precious stones on private land), as well as slate, talc, and all other substances obtained by mining. (Sec. 2.) In other words, the ordinance relates to all mines upon Crown land and to all mines on private land except those of gold, silver, and gems.

This ordinance, rather than giving detailed regulations, lists the matters concerning which the Governor in Council may formulate rules; it makes it imperative that any person (including any association or body of persons, incorporated or not) shall file a declaration of his intent to mine; and it provides penalties for the infraction of rules or regulations.

17/ Ordinance No. 2 of 1896, mines and machinery protection ordinance.

All rules, made by the Governor in Council, shall be laid before the Legislative Council within one month of the first session called after the making of the rules; and they shall cease to have effect if not approved within two months of their presentation to the Council. (Sec. 5.)

All rules, alterations, amendments, or cancellations thereof shall have effect only upon due publication in the Government Gazette. (Sec. 4.)

Subjects of Regulation

The Governor, with the advice of the Executive Council, shall make and when made alter, amend, or cancel rules with respect to:

1. Inspecting, examining, and ensuring the ventilation of any mine or part thereof.

2. The safety of employees in mine or factory and the fencing of machinery in or attached to any mine or factory.

3. Sanitary conditions in mine, factory, and surroundings.

4. Notices to owners, superintendents, managers, or persons in charge of mine or factory.

5. Appointing an Inspector or Inspectors of Mines and Factories.

6. Imposing restrictions on the cleaning of machinery while in motion.

7. Imposing restrictions on the employment of women and children between the fixed and traversing parts of any self-acting machine while it is in motion.

8. Reporting to the Government Agent of the Province (or the assistant government agent of the district) and to the Inspector of Mines and Factories, by the owner, superintendent, manager, or person in charge of any mine or factory, of loss of life or injury to any employee by reason of any accident or mishap at the mine or factory.

9. Holding inquiries and investigations with respect to such accidents or mishaps, enforcing attendance of witnesses, producing papers, and determining the persons by whom the costs of the inquiries and investigations are to be paid and the manner of enforcing such payment.

10. Any other matter necessary to the carrying out of the provisions of the ordinance. (Sec. 4 and 2A.)

Declaration of Intent to Mine

If any person intends to open, work, or use any mind, he shall, one calendar month before commencing to do so, furnish to the Government Agent of the Province in which the mine is situated a written declaration, containing the following particulars: (1) The name and boundaries of the land; (2) the nature of the right sought; and (3) the names and residences of himself and of those under whose management or superintendence the mine is to be opened, worked, or used. Any change made in such personnel shall be the subject of a further declaration. (Sec. 3.)

Penalties

Any mine operator failing to make the required declarations, committing a breach of the ordinance or of the rules made thereunder, obstructing the work of Government inspectors, neglecting or refusing to execute any written order, or keeping mine or factory in an unsanitary or poorly ventilated condition shall be liable to a maximum fine of Rs. 500 or to rigorous imprisonment for a maximum term of three months (or both) and upon a subsequent conviction to a maximum fine of Rs. 1,000 and imprisonment for a maximum term of six months (or both). (Sec. 6.)

A prosecution must be instituted within one year of the offense. (Sec. 7.) It shall be lawful to the court to award to the informer a share not to exceed one half of the fine actually realized. (Sec. 8.)

Gold, Silver, Gems, and Precious Stones on Private Land 18/

The Governor in Council may make and when made alter, amend, or cancel rules for (1) inspecting and examining into the state and condition and ensuring due ventilation of any mine or part thereof; (2) regulating all matters and things connected with the safety of employees; and (3) every other purpose necessary for carrying out the provisions of the ordinance: Provided that no rule, alteration, cancellation, etc., shall have effect until published in the Government Gazette.

SUPPLEMENT

Mining Industry

The developed or commercial mineral resources of Ceylon are confined practically to graphite, gems, slate, mica, and monazite sand, the most important being graphite, or plumbago. No mineral oils have been found, and geological surveys show no promise of their being discovered.19/ These

18/ Ordinance No. 5 of 1890, sec. 10.
19/ Leonard, Walter A., Mining Laws and Legal Restrictions Imposed on Foreigners in Securing or Operating Mineral Concessions: Consular Rept. 129632, Colombo, Oct. 25, 1919, Bureau of Mines foreign file 742.

minerals and a few others are briefly discussed in the following paragraphs, most of the facts being from a summary by Adams[20] and a report by Leonard.[21]

Iron ore.--The Singhalese, in earlier times, carried on the manufacture of iron and steel in all parts of the island, or at least in all the districts of the central part. Up to 1904, the industry was still active, upon a small scale, near Balangoda; but it gradually gave way to the cheaper iron and steel from Europe. The ore is hematite or limonite from shallow excavations in the lateritic soil; the iron was smelted from it in small furnaces of the "Catalan forge" type.

The deposits of titaniferous iron ore on the east coast, 40 miles north of Trincomalee, estimated to contain more than 5,000,000 tons, have not been opened up.

Thorianite.--Thorianite occurs in many parts of the island, underlain by the crystalline rocks of the Archean; it was found in the alluvial deposits in river valleys and in the beds of rivers. In the hope that they might be an economic source of thorium, uranium, and radium, attention was turned to these deposits. Expectations have not been realized, as no large deposits have been found.

Monazite.--Monazite sand has been known for some time to exist in Ceylon, although the mines have not been worked to any extent recently. In 1918 a special plant for the refining of the sand was erected; by the end of the year about 20 tons had been separated, ready for shipment to England. In 1921, 75 tons were exported from Ceylon; in 1922, 100 tons. In 1928, about 85 tons were exported. The Ceylon Blue Book for the year 1930 does not include monazite in figures of mineral production.

Mica.--In Ceylon phlogopite is associated with the crystalline limestones at several places; some muscovite also is said to exist in the island, but it is thought that only a small percentage of it is marketable.

Some mica was mined during the war period because of the great demand for the product in the United Kingdom. The latest Government report to include mica production is that of 1925,[22] which places output at 22 hundredweight, valued at Rs. 5,110. The Blue Book for 1931 reports a production of 31½ hundredweight, valued at Rs. 5,425, for 1930.

Salt.--Salt is made by the evaporation of sea water in shallow basins or lagoons. No salt is exported; production is almost adequate for domestic needs. Salt production in 1928, according to a Ceylon Government report, was 190,659 hundredweight, valued at Rs. 571,997. It is protected by a duty of Rs. 3 (97 cents) a hundredweight.

20/ Adams, Frank Dawson, The Geology of Ceylon; Minerals of Economic Value: Canadian Jour. of Research, vol. 1, No. 6, Dec., 1929, pp. 467-511.
21/ Leonard, Walter A., Work cited.
22/ Annual reports to the Bureau of Mines from the Government of Ceylon.

Graphite.--The graphite deposits of Ceylon are among the most important in the world, as measured by value of product. Mining and exporting began in 1824. Competition from Madagascar, together with the decrease in the demand for graphite crucibles, has made itself felt in Ceylon's output. In 1909 not less than 770 pits were being operated. According to Leonard,23/ the number of mines working at the beginning of 1919 was 263, and the number of men employed was 6,433, as compared with 1,288 mines in operation on June 20, 1917, employing 19,912 men, when the market was booming. In 1930, the number of active pits was 47; production amounted to 5,548 tons, valued at Rs. 644,315.

Plumbago mining is largely in the hands of the native Ceylonese. The majority of owners are men of small means, whose labor forces usually do not number more than 20. These owners rely on the immediate sale of their product; therefore, marked fluctuations in the market will cause them to shut down or open up their mines.

Gems.--Ceylon is one of the great gem-producing countries of the world. The gems are in alluvial deposits along the course of the present rivers or of those of former times. Aquamarine, amethyst, and moonstone are found in their original matrices. Other precious and semiprecious stones are chrysoberyl (and its varieties alexandrite and cat's eye), garnet (cinnamon stone), peridot, ruby, sapphire (which is the most valuable of the gems of Ceylon), spinel, topaz, tourmaline, and zircon.

The principal gem-producing districts are Balangoda, Rakwana, and Ratnapura, although gems are found in many other parts of the island. Lack of knowledge of the conditions that prevail in that country and the difficulty in securing title to any large alluvial areas have been the chief reasons for the failure of large companies or firms to enter the gemming business in Ceylon. The alluvial land is generally subdivided among peasant proprietors, who engage in the work only at certain seasons, as November, May, or June. To ascertain the value of the gems, the owners offer them at public auction, at which the gemmers themselves may bid and buy the gems if not satisfied with the prices offered by outsiders.

The value of the gem production in Ceylon, according to the Government mineralogist, is between $264,000 and $330,000 yearly, except when the discovery of an exceptional stone causes the value to be above normal.

A Government report for 1930 says that no accurate record of output is kept, as the gems do not pass through the customs office but are shipped by parcel post; however, in 1928 the Government reported that exports of corundum and gems were valued at Rs. 12,790. The 1931 Blue Book reports roughly a production in 1930 of gems valued at Rs. 98,517.50--all from the districts of Ratnapura and Kegalla, in the Province of Sabaragamu.

Miscellaneous.--The Blue Book of 1931 reports production in 1930 of cabook (building stone), granite, gravel, rubble, coral stones, and limestone.

23/ Leonard, Walter A., Work cited.

I. C. 6716

May, 1933

Department of Commerce

United States Bureau of Mines
Scott Turner, Director

INFORMATION CIRCULAR

MINING LAWS OF FRENCH EQUATORIAL AFRICA, WEST AFRICA, CAMEROUN, AND TOGO

BY

Paul M. Tyler

I.C. 6716
May, 1933.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING LAWS OF FRENCH EQUATORIAL AFRICA, WEST AFRICA, CAMEROUN, AND TOGO[1]

By Paul M. Tyler[2]

FOREWORD

This paper presents one of a series of digests of foreign mining legislation and court decisions which is being prepared in advance of a general report relative to the rights of American citizens to explore for minerals and to own and operate mines in various foreign countries. This interpretation of the mining laws of specified French possessions in Africa has been prepared from the best information available in Washington, but is released subject to correction and amplification, if necessary, by the proper American diplomatic and consular officers, to whom it is being referred through the courtesy of the Department of State.

France controls large areas on the African continent, but the various Colonies and Dependencies are administered more or less independently. Algeria is under the direct jurisdiction of the Ministry of the Interior, and its mining laws are essentially the same as those of the mother country. Tunis and Morocco are attached to the Ministry of Foreign Affairs; separate digests of the mining laws of these respective colonies have already been prepared by the United States Bureau of Mines. The present paper covers all other French territory on the continent of Africa except Somaliland, a rather small Colony including the port of Djibouti near the point where the Red Sea joins the Gulf of Aden. Available information indicates that the prospects of mineral production in this Colony, apart from salt and gypsum, are not at all encouraging.[3]

INTRODUCTION

The largest of all French possessions, eight times the area of France itself, is French West Africa, which includes some $1\frac{1}{2}$ million square miles of the Sahara Desert, along with various contiguous territories extending to the

1 The Bureau of Mines will welcome reprinting of this article, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6716.

2 Chief engineer, rare metals and nonmetals division, U. S. Bureau of Mines.

3 Dreyfuss, M., La Geologie et les mines de la France d'outre-mer: Paris, 1932, p. 290.

coast. The approximate population of this vast region is around $14\frac{1}{4}$ millions, of which in 1926 only about 13,500 were non-African races. There is a Governor-General at Dakar but the direct administration of each of the nine component Colonies is under a local Lieutenant-Governor. The following Colonies comprise the group: (1) Senegal, (2) French Guinea, (3) the Ivory Coast, (4) Dahomey, (5) French Sudan, (6) Upper Volta, (7) Mauritania, (8) Niger, and (9) Circle of Dakar and Dependencies. Properly speaking there is no Mines Service in French West Africa. Its functions are performed at Dakar by the Mines Section of the Inspection General for Public Works and in the various Colonies by the mining engineer attached to local Public Works Services. The Geological Service, on the other hand, was centralized at Dakar in 1930, field work being done in the dry season and office work at Dakar during the wet season.

Togo and Cameroun are French mandates (formerly German) under the French Minister of the Colonies. The former is administered by a Commissioner and the latter by a Governor; each has its own Mines Service.

French Equatorial Africa or French Congo comprises a group of four Colonies (Middle Congo, Gabon, Oubangi-Chari, and Tchad) under a Governor-General. The total area is about 14 times that of the mother country. The Mines Service has its headquarters at Brazzaville. The geologists report to the Mines Service.

In Somaliland, which is administered by a Governor, a geologist is attached to the Service for Public Works and Mines.

MINERAL RESOURCES

The various Colonies of French West Africa and Togo are reported[4] to contain deposits of bauxite, bismuth, bitumen, chromite, copper, gold, gypsum, iron, lead, limestone, manganese, mica, phosphate rock, quartz, tellurium, tin, titanium, tungsten, and zircon; but gold (much of it mined by natives), ilmenite sands (at Dakar), and salt are the only minerals commercially produced. The chromite deposits in Guinea and Togo are probably of commercial importance, but the manganese deposits of French Sudan, although similar to those of the Gold Coast, are too inaccessible, being many hundreds of miles from the coast.[5]

In French Equatorial Africa and Cameroun, copper ores have constituted the only commercial mineral product. Lead, zinc, and silver accompany the copper in Niari and Djoue deposits, but in minor amounts. Graphite has been

4 Jones, L. M., Mineral Production of the World, 1924-1929: Min. Resources of the United States, Part 1, 1932, p. 897.

5 Hubert, H., La Geologie et les mines de la France d'outre-mer: Chap. IV, Afrique occidentale française et Togo. Publication of the Bureau d'études géologiques et miniéres coloniales, Paris, 1932, pp. 205-239.

noted near Franceville and in Oubangi-Chari, where alluvial diamonds are likewise reported. Iron ore is available in several localities, but coal has not been found, being absent in the extrusions of the strata (Karroo system) from which it is obtained in the Belgian Congo. Bitumen has been discovered and somewhat favorable indications of petroleum have been developed in prospecting.[6]

SOURCES OF LAW

General observations on French colonial legislation with the titles of the principal mining laws for each Colony have been compiled by a special committee and published recently in Paris.[7] The present paper has been prepared by abstracting and comparing the original texts as published in the Journal Official of the French Republic, and all references herein, unless otherwise noted, are to the basic decrees for the respective Colonies, namely:

French West Africa, October 22, 1924 (subsequently modified, with respect to reserved zones, by decree of July 31, 1927).
Togo, November 26, 1927.
Cameroun, May 20, 1928.
French Equatorial Africa, July 8, 1926 (subsequently modified, with respect to reserved zones, by decree of July 8, 1926).
Somaliland, July 6, 1899 (subsequently modified, with respect to reserved zones, by decree of March 13, 1928, which also rendered applicable to this Colony the decree of July 31, 1927).

All French possessions in Africa except Algeria and Tunisia were formerly covered by the mining law decree of July 6, 1899. In the case of French West Africa the reasons given for issuing a new decree were chiefly to ratify local customs as regards gold prospecting and to encourage the search for oil. In transmitting the new decree with respect to French Equatorial Africa, the Minister of the Colonies mentioned the desire to avoid monopolies and inadequate exploitation under concessions and to assure for the Colony an equitable share of the proceeds of mining. With respect to Togo special attention was given to the protection of native labor, and the law for Cameroun differs from that of Togo only as regards preexisting mining rights.

The basic decrees for all the Colonies except Somaliland are drafted on so nearly the same plan that the numbering of the articles is in many cases identical. Where there is a slight difference in numbering, the present paper follows the sequence of the law for French West Africa.

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6 Demay, A., La Geologie et les mines de la France d'outre-mer: Chap. V, Afrique équatoriale française et Cameroun, Paris, 1932, pp. 241-284.
7 Comité d'études minières pour la France d'outre-mer: Annuaire, Paris, 1932, 376 pp.

CLASSIFICATION

Mineral deposits are divided into quarries and mines. (Art. 2.) Quarries comprise all deposits of building materials, fertilizers, and similar substances except nitrates and associated salts and phosphates. Peat bogs are classed as quarries. (Art. 3.) In French Equatorial Africa, deposits of certain substances may, according to their ultimate use, be classified simultaneously as quarries and as mines. Questions as to proper classification may be decided by the Minister for the Colonies with the advice of the Colonial Committee for Public Works. (Art. 4.) The title to deposits classed as "quarries" is not separated from the surface rights, and quarries consequently do not come within the province of the mining laws except as regards inspection and other measures for preserving the security of the surface and the safety of workers.

All minerals not classed as quarries are classed as mines. In French West Africa, Togo, and Cameroun, mines are divided into three categories as follows (art. 6):

1. Precious metals and precious stones.
2. Oil, gas, bitumens, and oil shale.
3. All other substances.

In French Equatorial Africa, precious metals and precious stones are not separated, leaving only two categories.

Questions of legal classification are decided by the Governor (French Equatorial Africa), Commissioner (Togo and Cameroun), or the Governor-General (French West Africa).

OWNERSHIP AND PROPERTY RIGHTS

Quarries belong to the owner of the land. Right to exploit mines is acquired from the State in accordance with the provision of the mining law, following the issue of an exclusive prospecting permit. Subsoil rights for minerals of different categories may be accorded to different persons for the same area.

Prospecting permits in several French Colonies are classed as real property, but in the areas under discussion they are described as personal property. A prospecting permit may be transferred but cannot be divided or mortgaged. (Art. 10.)

A mining concession constitutes real property distinct from ownership of the soil but of limited duration. (Art. 10.) Except that it may not be divided and is subject to certain other restrictions of the mining law, it is conveyed like any other form of real estate. (Art. 12.)

PERSONAL LICENSES REQUIRED

In accordance with general legislation extending to all but three of the French Colonies (decree of February 27, 1924), no individual or company may undertake to prospect or exercise rights under a prospecting permit or mining concession without first obtaining a personal license (autorisation) from the Lieutenant-Governor, Commissioner, or corresponding administrative authority. (Fee, 100 fr.) The personal license is subject to cancellation by the issuing official, who need not give his reasons therefor; but such cancellation cannot be made retroactive. (Art. 14.)

RIGHTS OF FOREIGNERS

The provisions of the war-time decree of January 8, 1916, are continued (art. 15); these include the requirement that all companies formed for prospecting or mining must conform to French laws and have their main office in France or in French Colonies. Three fourths of the directors of corporations, including the president and active executives, must be French. Exception is made in Togo or Cameroun (mandates), however, where privileges are extended to citizens of the United States and to those of all countries adhering to the League of Nations.

All mining companies must furnish a copy of their bylaws and list of members, administrative officers, directors, etc., and must keep the authorities informed of all subsequent changes in the organization or management. Sundry provisions in the laws assure complete knowledge of the nationality of all prospectors and mining companies and their staffs and of those who gain control of mining or prospecting rights through inheritance, purchase, gift, or foreclosure.

NATIVE MINING RESERVATIONS

In the basic decree for French West Africa, article 17 provides for continuing the customary rights of natives to obtain gold and other mineral substances in districts traditionally exploited by them. No such provision appears in the laws of the other territorial units discussed in this paper. Such rights are not transferable to Europeans and are not enjoyed by natives who are not indigenous to a particular district. Under certain circumstances, however, prospecting permits and concessions may be issued for areas in these native reservations, usually subject to continuance of native rights or indemnity therefor. (Art. 46.)

PROSPECTING PERMITS

General.- A prospecting permit confers exclusive rights to search for deposits of the specified category within a designated area (subject in French West Africa to the rights of natives). Priority of registration (with the Commandant of the Circle or the Chief of the "Circumscription") carries with

it a prior right to receive a permit as against other applicants (arts. 20, 25), but application in all cases must be preceded by actual location on the soil of a substantial marker either at the center or corners or, in the case of a dredging claim (in French West Africa only), at all four corners. To these markers must be affixed the date, name of the applicant, and the category of mineral found. (Art. 21.)

Area.- The claim is in the form of a square whose sides are oriented true north and south and east and west. In French Equatorial Africa, each side is 10 km. long; elsewhere it is 3 km. In French West Africa, there is a special provision for a dredging claim following the bed of a stream bounded by parallel side lines and not more than 100 m. on each side from the center line of the stream; the end lines are perpendicular to the center line, and the area is not less than 100 nor more than 400 hectares. (Art. 20.)

If the boundaries of the claim as prescribed overlap an area previously granted for mining or prospecting minerals of the same category, the new claim shall be reduced by the extent of such overlap. Similar reductions are provided in case of overlap with restricted or reserved areas. (Art. 20.)

Application.- The application should be delivered to the office of the Commandant of the Circle by the applicant or his accredited agent. However, it may be sent through the mails, although at the risk of the sender. If the claim overlaps two circles, a copy of the application should be sent to the commandant of the other circle also. (Art. 23.)

The application should set forth (1) the full name, title, nationality, and usual domicile of the applicant, together with his chosen address in the colony or that of his representative, or, with respect to a company, the name, main office, and the full name and address of its resident agent; (2) the date when the location marker was set up and the wording of the notice thereon; (3) the situation of the area applied for; (4) the category of mineral and the name of the mineral actually discovered.

Separate application must be made for each area and for each category of mineral applied for.

To the application should be attached (1) documentary evidence with respect to the eligibility of the applicant (as defined in arts. 14, 15, and 16); (2) a map of the country with the desired area indicated thereon; and (3) a map or sketch (croquis) of the surface on a scale of 1:10,000, oriented to the true north and indicating the positions of the center or the corners of the claim (in the case of a dredging claim for at least one of the corner stakes) with reference to one or more permanent landmarks easily recognized in the field; and (4) a receipt for the deposit of the required fee. (Art. 24.)

In French Equatorial Africa where the claim is large the fee is 500 francs; elsewhere it is 100 francs per claim. (Art. 22.)

After registration the application is forwarded to the Chief of the Mines Service, who cannot reject it except for the following reasons (Art. 26):

(1) If the application contains serious errors that cannot be corrected or if it fails to conform with the conditions specifically imposed in the mining law;

(2) If the area itself is found to be situated entirely (a) within an area already claimed with respect to minerals of the same category of (b) within a reserved area or district;

(3) If the interested party fails to furnish the required information within a reasonable time.

Right of appeal (to the Lieutenant-Governor or corresponding official) is granted (without prejudice to subsequent court proceedings) within one month after denial by the Chief of the Mines Service. (Art. 27.)

Duration.- Except in French Equatorial Africa, where it is good for 2 years, a prospecting permit is valid for 3 years (art. 28), and in all the regions herein discussed it may be renewed two or more times for additional periods of 2 years each, even though a change in ownership has taken place. The fee for the first renewal is 200 francs and for the second renewal 300 francs (or double and triple the original fee if over 100 francs) except in French Equatorial Africa where the first renewal costs 500 francs and the second 1,000 francs (or equal to and double the original fee if over 500 francs). (Art. 29.)

A prospecting permit becomes null and void when it expires, and unless a concession has been applied for the area becomes open again for prospecting. (Art. 30.)

Disposal of product.- Prospecting work which degenerates into exploitation is forbidden and subject to punishment by fine and imprisonment as illegal mining. (Art. 36.) Nevertheless, a prospector may freely dispose of concessionable minerals produced by his labors, subject to the payment of taxes and royalties for substances of like character. He also must notify the Chief of the Mines Service, who furnishes a certificate of permission to ship. This authority is valid for 1 year and may be renewed. (Art. 34.)

Cancellation.- Provision is made for withdrawal of a prospecting permit for specified infractions of the mining laws, including failure to report within one month the amount of production. (Art. 35.)

CONCESSIONS

General.- The right to exploit mines is acquired only by obtaining a concession, and a concession can be applied for only by the owner of a valid prospecting permit covering the same category of minerals in the same area. (Art. 5.)

A prospecting permit which has not expired carries with it the right to obtain a concession. (Art. 38.) Application therefor must be addressed to the Chief of the Mines Service at least 30 days before the permit expires and must be accompanied by the fee of 500 francs -- or 1,000 francs in French Equatorial Africa. (Art. 40.) This application should contain the same essential information as that for a prospecting permit as regards the identity of the applicant, the mineral, and the description of the area, together with the number of the prospecting permit and supporting evidence relative to the nature and characteristics of the deposit found. (Art. 40.) Registration of the application may be refused only for failure to pay the required fee; and if the prospecting permit should expire before the concession is granted or denied, an extension is automatically provided. (Art. 41.) If the application is in due form, it is forwarded by the Mines Service to the Lieutenant-Governor (or similar authority) and it is advertised three times in the Journal Officiel at intervals of 15 days or more, a total of at least 3 months being allowed to elapse after the first insertion, during which time an investigation is made by the Mines Service and the local authorities (Commandant of the Circle). The cost of such investigation is borne by the applicant. (Art. 44.) In the absence of effective opposition from other interested parties, the Lieutenant-Governor (or similar authority) issues the concession subject, however, to a check up on the boundaries within 6 months (Art. 49) and a final review after 3 years by a committee of three members, one of whom represents the Mines Service. (Art. 46.) This committee considers whether the concessionnaire has performed enough work to represent a normal exploitation of all concessions belonging to him and thereupon decides whether (1) to cancel the concession, (2) to extend it for another probationary period (not more than 2 years), or (3) finally to ratify the concession. (Art. 57.)

Area.- The unit area of a concession is a rectangle oriented due north and south and wholly within the area covered by the prospecting permit or (outside of French West Africa) included within prospecting permits owned by the applicant. In French Equatorial Africa the area is not further limited, but elsewhere it may not exceed 900 hectares and the shorter side of the rectangle must be at least one fourth as long as the longer side. Dredging claims in French West Africa must conform to the conditions noted for prospecting permits (see p. 6, Area) and may not exceed 400 hectares each. (Art. 38.)

As in the case of prospecting permits, if the area claimed overlaps areas previously granted for mining or prospecting minerals of the same category, the concession shall be reduced by the amount of such overlap. Similar reductions are provided in case of overlap with restricted or reserved areas. Special conditions covering native mining reservations are discussed in the French West African law. (Art. 46.)

Fusion of two or more concessions may be permitted at the discretion of the authorities but division is prohibited. (Arts. 51, 52.)

Duration.- The period of a concession is 50 years (75 years in French Equatorial Africa) and it may be renewed two or more times for additional periods of 25 years, provided the property has been worked with sufficient activity. Application for renewal should be addressed to the Chief of the Mines Service at least 3 years, before the original concession expires. (Art. 39.)

Annulment.- The validity of a concession is always subservient to the existence of another valid prospecting permit or concession based upon a prospecting permit of prior issue. (Art. 48.) Failure to pay taxes and other charges may result in forfeiture (art. 56) and forfeiture may follow violation of provisions surrounding transfer of the property or the laws governing the composition of corporate enterprises. (Art. 58.) A concession that is not kept in operation may be cancelled. (Art. 59.)

RENTS AND ROYALTIES

A fixed surface rental of 2 francs per hectare increasing to 4 francs a hectare after 10 years (somewhat different in French Equatorial Africa) is charged on all concessions (art. 54), and in addition a royalty of 5 percent is charged on the value of the product at the mine. In the case of mineral produced in prospecting, the rate is 7 percent. (Art. 55.) In French West Africa, the basis of royalty may be changed after 1 year at the request of the concessionnaire to 10 percent of the net proceeds; in the other Colonies this change is to 15 percent and at the option of the Government.

RELATIONS WITH LANDOWNERS AND OTHER MINERS

Walled enclosures, streams, and gardens cannot be occupied without formal consent of the landowner, and without the latter's permission no mine openings can be made within 50 meters of a dwelling or adjoining land. The laws and customs respecting graves must also be observed. (Art. 60.)

On the free public domain, a concessionnaire may freely occupy within the limits of his concession all land necessary for his prospecting, mining, and the mechanical preparation of the product and also for ditches, canals, and communications, as well as markers and monuments -- subject to approval of the Governor. He may use any water not already utilized and cut timber necessary for his mining operations. Moreover, he has a prior right to acquire the surface rights for the same area. (Art. 62.)

If the land is occupied by natives, an annual rental must be paid for its use. If the use of the land for mining purposes tends to monopolize the land for more than 1 year or if it occasions destruction of crops or of the agricultural value of the land itself, it may be necessary to purchase the

land at a price which, in default of amicable agreement, shall be fixed by the Lieutenant-Governor. (Art. 63.)

Private lands may be occupied for mining purposes in default of agreement with the owner only after authorization from the Government after a hearing with the proprietor and with the advice of the Chief of the Mines Service. The amount of indemnity may be decided in the courts. In the case of temporary occupation the damages amount to double the net yield of the land, but if the landowner is deprived of the use of his land for more than a year, if crops or trees are destroyed, or if the usefulness of the land is impaired, the land must be purchased outright at double its value at the time of occupation. (Art. 64.) Rights of way and other privileges outside of the concession are discussed in articles 65 to 67. Damage to adjoining mines must be recompensed (art. 69), but neighboring owners may not oppose work necessary for drainage, ventilation, or safety work (art. 68), and if the neighboring mine benefits thereby compensation can be claimed. (Art. 69.)

Holders of prospecting permits have similar rights to those of concessionnaires except with respect to acquiring ownership of surface rights. (Art. 76.)

MINE INSPECTION

The usual provisions in French colonial law for supervision of mining operations by the Mines Service to assure health and safety of workers and protection of the public interest are contained in articles 69-75.

JURISDICTION AND PENALTIES

Appeal from decisions of the Lieutenant-Governor may be made before the Council of the State. (Art. 77.) Cases involving overlap of claims are decided by the courts, advice from the Mines Service being entered as expert testimony. (Art. 78.) Charges of violations of the mining law may be brought by the judiciary police, agents of the Mines Service, and all other agents commissioned for this purpose by the Governor. Such charges constitute proof in the absence of proof to the contrary and opposition thereto must be entered within 30 days. (Art. 79.)

A fine of from 1,000 to 25,000 francs <u>and</u> imprisonment for from 3 months to 3 years are provided for illegal exploitation of precious stones or precious metals, and the product is confiscated. (Art. 81.)

A fine of from 100 to 1,000 francs or imprisonment for from 15 days to 2 years, or both, is provided for (1) false testimony regarding the setting up of a claim marker, (2) willful destruction or tampering of claim monuments, (3) falsification as to date of a prospecting permit, and (4) false declaration of identity or misrepresentation regarding other essential matters in order to obtain a prospecting permit. (Art. 82.)

Minor infractions of the mining code are punishable by fines of lesser amount, and imprisonment up to 1 year also may be ordered in case of illegal exploitation of deposits of other than precious metals or stones. (Arts. 83 and 84.)

For a second offense the maximum fine and imprisonment must be imposed and it may be doubled. (Art. 85.)

Those who have been imprisoned for infractions of the mining code become ineligible to obtain prospecting permits or concessions for 3 years, and existing permits may not be renewed for a similar period. (Art. 87.)

MISCELLANEOUS PROVISIONS

The Lieutenant-Governor (or similar authority) with the advice of the Mines Service may promulgate regulations necessary for carrying out the provisions of the mining law. (Art. 97.)

The Mines Service (as organized under the decree of August 5, 1910) is charged with the administration of the mining laws. (Art. 98.)

For the public interest the Governor-General in Privy Council may issue an order suspending for 2 years the right to obtain prospecting permits in designated regions. (Art. 99.) These powers with respect to the Equatorial and West African Colonies were further strengthened by a decree dated July 31, 1927. The Governor-General may also requisition for Government use any mineral produced, subject to suitable payment (which may be settled by the courts). (Art. 100.) He is further empowered to prohibit fusion of two or more mines under the ownership of the same individual or corporation, if such fusion appears contrary to the public interest. (Art. 101.)

Notice of the institution of prospecting permits or concessions as well as all changes, court decisions, etc., are registered in accordance with the laws relating to real estate. Applications for and deliveries of prospecting permits and concessions and virtually all official business relating thereto is advertised in the Official Journal of the Colony.

Government employees, military officers, and soldiers in active service are forbidden to take an active interest in prospecting or mining; personal license will not be accorded them. Employees of the Mines Service are forbidden to take even an indirect interest. (Art. 13.)

The exploitation of mines is considered an act of commerce. (Art. 18.)

OIL AND GAS

The provisions of the general mining law are all applicable to substances of the fourth category except as provided for in articles 90 to 96 (and subsequent legislation). The fees for prospecting permits and renewals thereof are larger, and a second renewal is contingent upon the performance of certain minimum working requirements. (Arts. 91 and 92.)

Provision is made for group development, work done on one claim being credited against the requirement for all the claims in the group. As in other French Colonies more credit is accorded for deep holes than for shallow holes. (Art. 93.) Royalties are reduced to 2.5 percent during the first 5 years for each of the first 10 concessions of this category which attain an annual output of 5,000 metric tons. (Art. 96.)

EXCLUSIVE PROSPECTING RIGHTS

By a Presidential decree dated May 14, 1930, exclusive general prospecting rights for 5 years were granted in French Equatorial Africa to the Oubangui Exploration Co. (Cie générale de recherches minières de l'Oubangui) covering all minerals classed as "mines," except oil and gas, over a huge tract, subject to any previous existing rights. The elaborate contract includes sundry provisions to assure that any subsequent concessions shall be actively worked.

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

MINING LAWS OF SURINAM (DUTCH GUIANA)

BY

E. P. YOUNGMAN

I.C. 6717
May, 1933.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE--BUREAU OF MINES

MINING LAWS OF SURINAM (DUTCH GUIANA)[1]

By E. P. Youngman[2]

PREFATORY NOTE

This paper is one of a series of digests of foreign mining legislation and court decisions that is being prepared in advance of a general report relative to the right of American citizens to explore for minerals and to own and operate mines in various foreign countries. This interpretation of the mining laws of Surinam, or Dutch Guiana, is based primarily upon translations[3] of copies of laws forwarded by John V. Swearingen, American vice consul, Georgetown (British Guiana), and submitted to the Bureau of Mines through the courtesy of the Department of State. This digest is released subject to correction and amplification, if necessary, by American foreign-service officers.

INTRODUCTION

The basic mining legislation of Dutch Guiana is comprised in the law of September 7, 1882 (concerning mining in general), and the decree of December 1, 1894 (concerning mining in navigable streams and in creeks), together with the following supplementing or amending acts:

No. 1 of 1905 (Jan. 3, 1905), referring to the text of the order of Sept. 7, 1822, with respect to exploration work in Surinam.

No. 2 of 1905 (Jan. 4, 1905), referring to the insertion in the official organ of the Government of the text of the order of Dec. 1, 1894, concerning work in navigable streams and in creeks.

No. 31 of 1908 (March 18, 1908), containing additional modifications of the order of Sept. 7, 1882, pertaining to conditions for exploring and mining.

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1 The Bureau of Mines will welcome the reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6717."

2 Rare metals and nonmetals division, U. S. Bureau of Mines.

3 Orrmins, William R., translator.

No. 32 of 1908 (March 18, 1908), containing additional modifications of the order of Dec. 1, 1894, pertaining to conditions for exploring and mining in navigable streams and in creeks.

No. 61 of 1916 (Aug. 30, 1916), supplementing the order of Sept. 7, 1882.

Translations of these acts (with the exception of No. 61 of 1916) were not available in the preparation of this paper. However, the summary of mining legislation in Dutch Guiana made by Van Wagenen,[5] which includes the law of 1882, the law of 1894, and all supplementary legislation up to January 1, 1917, has been extensively used in the discussion of general mining legislation. As the mining of gold (the only mineral except bauxite mined in commercial quantities) is entirely from alluvial workings, the old legislation is probably adequate with reference to this mineral. The laws (passed in 1919 and in 1924) governing bauxite and petroleum and coal have been considered separately.

The recent production of gold and bauxite is shown in the following table:[6]

	1921-25	1926	1927	1928	1929	1930
Gold ... troy ounces	11,272	8,327	7,716	5,498	3,569	4,758
Bauxite . metric tons	[1]43,130	46,500	181,600	213,900	210,000	264,555

1 Four-year average.

According to the latest available information, no oil has been produced in Surinam, although search therefor has gone on intermittently since 1924. A prosperous oil industry has been predicted for the colony. Extensive new oil lands, with the same surface conditions as those of the great oil-producing regions of Venezuela, stretch westward from New Nickerie to the east bank of the Corentyne River, opposite Springlands (on the British Guiana bank).

RIGHTS OF FOREIGNERS

"Only citizens of the Netherlands or of the colony of Surinam or companies legally organized under the law of either country and represented in Surinam by a legally authorized agent may acquire mining rights in Surinam.[7]

4 The original texts of the supplementing acts, but not the text of the 1882 and 1894 laws, are in the files of the U. S. Bureau of Mines.

5 Van Wagenen, Theo. F., International Mining Law: McGraw-Hill Book Co., 1918, pp. 257-260

6 Commerce Yearbook, 1931, Surinam (Dutch Guiana): Commerce Yearbook for Foreign Countries, vol. 2, No. 9, 1931, p. 429.

7 Van Wagenen, Theo. F., Work cited.

The recent laws, the bauxite law of 1919 and the coal and petroleum law of 1924, have the following provisions:

> Holders of bauxite prospecting permits shall be (1) Dutchmen, (2) residents of the Netherlands, (3) residents of Surinam, (4) companies established ("residing") in the Netherlands, and (5) companies established in Surinam.
>
> Holders of bauxite mining concessions shall be companies established in either the Netherlands or Surinam. (Sec. 1 and 2, art. 2, No. 80 of 1919.)
>
> Differences of opinion concerning requirements of nationality and residence shall be decided by the judge, according to the manner fixed in a decree of April 1, 1903. (Art. 1, No. 79 of 1920.)
>
> Coal or petroleum permits and concessions, as well as transfers thereof, shall be granted only to (1) Dutchmen, (2) residents of the Netherlands, (3) residents of Surinam, (4) companies established in the Netherlands, or (5) companies established in Surinam. (Secs. 1 and 2, art. 3, No. 27 of 1924.) A permittee or a concessionaire not residing in the colony must be represented there. (Sec. 3, art. 3, No. 27 of 1924.)

The use of the word "residents" makes misunderstanding possible. A decision with respect to another Dutch territory (Curacao, West Indies) by a director of public works (through whose office pass applications for concessions) was to the effect that "residents" should be construed as synonymous with "Dutch subjects" and that a foreigner that had established his legal domicile in Curacao would be considered entitled to acquire a mining concession or to act as a director in a Dutch or Curacao corporation applying for a concession.[8]

Possibly facts speak better than the text of the law concerning foreign participation in mining.

The Surinam Bauxite Co., the only bauxite-producing company, is controlled by the Republic Mining & Manufacturing Co., which is a subsidiary of the Aluminum Co. of America. The property of the company includes a grant of 73,256 hectares (180,000 acres) on the east bank of the Cottica River, at Moengo, 100 miles southeast of Paramaribo. Deposits occur at a number of other points in the colony, 95 per cent of which is controlled by the company.[9]

With respect to petroleum, at the time the petroleum and coal act (article 2 of which gives the Governor authority to grant a concession to <u>one</u>

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8 Voetter, Thomas W., Status of British Oil Companies Operating in Dutch-controlled Territories: Consular Rept. 325351, Curacao, Dec. 19, 1929.

9 The Mines Handbook, Dutch Guiana: Vol. 18, pt. 2, Mines Information Bureau (Inc.), New York, 1931, p. 2845.

person to exploit any or all the petroleum and coal in Surinam) was passed, B. Koker, of Holland, received an exclusive concession and incorporated a limited liability company, the Surinam Oil Co. (Ltd.)[10] Koker, whose concession would have expired in the fall of 1929, died in June of that year, but it was not known what effect his death would have upon the concession, which was on the Nickerie shore, Nickerie district. Drilling, however, had been started on the Pln. Geyersvlyt, about 3 miles north of Paramaribo, on the Surinam River. This well was on private property (having no connection with Koker's concession) belonging to I. Hass, a local agent for Gillespie & Co. (of London).

An official Surinam Government report for the year 1930, made to the U. S. Bureau of Mines under date of May 31, 1931, is as follows:

> "The "Surin" Oil Co. did not continue to drill for oil,[11] nor did it make any search for this mineral. During the year a new exploration permit was granted to S. M. Filipovitch, an American engineer, on behalf of an American financier, for an area of 37,000 hectares.

GENERAL MINING LEGISLATION

Prospecting Permits

Prospecting is not free except to the landowner on his own land, and then only upon due notice given to the Superintendent of the Crown domain. When any one other than the owner of private land desires to prospect, he must obtain a written permit from the Governor with respect to Crown land and written consent from the owner with respect to private land.

A prospecting permit gives the holder the right at any time within the term thereof, or within an extension of that term, to select and stake off c part or the whole of the prospecting area and apply for the right to mine thereon.

A permit gives its holder the right to prospect only; itsgives no authority to remove or realize upon any metals or ores found. Exploration shafts and other excavations may be made freely, and drill holes may be sunk. Assay samples may be removed.

Application.--Every application for prospecting rights must be accompanied by: (1) The name, nationality, and legal residence of the applicant; (2) a map showing the relative position of the ground desired, as well as its area; and (3) a receipt from the Colonial Treasurer, showing that a sum equal to 1 cent a hectare per annum has been paid for the term desired.

10/ Smith, Gaston, Prospecting for Oil in Dutch Guiana: Consular Rept. 156880, Georgetown, British Guiana, Dec. 8, 1924.

11/ A similar report for 1929 stated that the Surin Oil Co. which until then had been the only holder of an exploration permit for oil, was continuing its search for this mineral.

The application must be signed by all parties at interest or by their duly authorized agents or in the case of a partnership or a company by its legal representative.

Authority.--The right to reject any application for reasons that to him seem sufficient is vested in the Governor, sitting in Privy Council.

Area.--A prospecting area may not be less than 200 or more than 20,000 hectares.

Duration.--The maximum term is three years, with the right of two renewals of one year each, making a total of five years.

Registration.--A prospecting permit is not valid until it is registered. The document (permit) must be presented to the commissary of the district in which the land to be prospected lies, who shall visit the property, verify the description thereof, and register the permit.

Transfer.[12]--The right held under a prospecting permit may be transferred (in whole or in part) under the written consent of the Governor, the transferee to be furnished with a new permit for the unexpired period of the original one.

Right to transfer shall be refused if the territory to be transferred or the portion remaining is less than 2,000 hectares. Transfer shall likewise be refused should the transferee thereby obtain the right to an area of more than 20,000 hectares.

Mining Concessions

The right to mine (priority to which belongs to the holder of a prospecting permit) when once granted conveys all the usual rights and privileges pertinent to the mining business, together with the right to engage in agriculture on the premises to the extent of raising thereon food for the consumption of the concessionaire and his employees.

Application.--Application for a mining right must be made in writing to the Governor; it must be accompanied by a receipt from the Colonial Treasurer, showing that a sum equal to the rental of the chosen tract for the first year has been paid.

Before application is made to the Governor, however, a provisional notice of the application contemplated must be given to the Superintendent of Crown lands. This notice must be accompanied by a map of the land desired, which must be prepared and sworn to by a Crown surveyor. The notice must be registered, and it must be filed with the Governor within two days of its registration. The street and number of the domicile (which must be in Paramaribo) of the applicant, or of his legal representative, must be given.

12 Decree No. 61 of 1916, substituting a new clause for article 4 of the decree of Sept. 7, 1882.

Authority.--The Governor, acting with the Privy Council, is empowered to reject an application in whole or in part, in which case the rental paid is in whole or in part returnable to the applicant. If the applicant is not satisfied with the decision of the Governor, he may withdraw his application completely and recover the full amount of the rental deposited.

Area.--No mining area shall be less than 200 hectares.

Duration.--No mining right shall be issued for less than one year or for more than 40 years.

Rent.--Rentals, which are payable annually and at least 30 days before the end of each year, shall be as follows:

Ten cents a hectare per annum for the first and second years.
Twenty-five cents per hectare per annum for the third and fourth years.
Fifty cents per hectare per annum for the remaining years.

If, at the option of the holder of a mining right, the area has been reduced at the end of any year of the term (by written application to the Governor made at least 40 days before the end of the year), a corresponding reduction of the rental for the next year shall be made. (But no reduction of area to a tract of less than 200 hectares is permitted.)

Failure to pay the required rentals automatically terminates the mining franchise.

Royalty.--No royalties or other dues of any kind beyond the rental are imposed. However, all gold recovered must be declared and sold to the Government, which pays for it the standard price in coin or currency, less a moderate charge for smelting, refining, and assaying.

Transfer.--A mining concession may be sold with the consent of the Governor, who with the sanction of the Privy Council may refuse the transfer right. A fee of 2 per cent of the amount named in the deed is collected by the Government on every conveyance.

Miscellaneous Labor Provisions

The employment of labor, either for prospecting or mining operations, must be conducted in the presence of the Commissary of Police, in accordance with the rules and regulations governing the employment of native labor. Labor may not be contracted for outside of the colony; nor may resident British Indian immigrants be employed in mining work.

Every laborer whose employment is accepted by the employer and the Commissary of Police must be registered by name; the amount of wages to be paid to him must be agreed upon, as well as the term of the employment and the locality in which he is to work.

The employer is responsible to the Government for the health, good treatment, subsistence, proper shelter, and wages of every laborer.

PETROLEUM AND COAL LEGISLATION

General

The law of May 20, 1924 (No. 27 of 1924), concerning the exploration and exploitation of petroleum and coal in Surinam, in article 1, provides that no rights of investigation or mining with respect to petroleum and coal are to be granted under the basic mining law of September 7, 1882 (modified and supplemented by a decree of August 30, 1916), or under that of December 1, 1894, concerning the exploitation of minerals in navigable creeks and streams (modified and supplemented by a decree of March 18, 1908). All rights with respect to oil and coal shall be obtained exclusively under the law of May 20, 1924, (Art. 1, No. 27, of 1924.)

The Governor shall have the authority to grant the right of determining the presence of, as well as the right to exploit, oil and coal in the domains of Surinam, in their entirety or in part, to one person. (Art. 2, No. 27 of 1924.)

Prospecting Permits

A prospecting permit entitles the holder, free of charge, to search for minerals on the lands stipulated, on condition that, when in the judgment of the Governor prospecting would be detrimental to the country or to third parties, no investigations shall be carried on until compensation for damages shall have been made or assured to the parties interested, according to regulations laid down by decree. (Sec. 2, art. 4, No. 27, of 1924.)

A prospecting permit is issued for a fixed period not to exceed five years. (Sec. 1, art. 4, No. 27 of 1924.)

Mining Concessions

The holder of a prospecting permit has the right to obtain a concession for the exploitation of coal or petroleum in that part (or parts) of Surinam for which he makes request. He shall accompany his application by a map, in duplicate, made by a certified surveyor. Application must be made at least six months before the expiration of the term of the prospecting permit. (Sec. 1, art. 5, No. 27 of 1924.) After having presented his petition for a concession, the permittee may continue his prospecting until he shall have been informed of the decision with respect to his application, even though the term of his prospecting permit may have expired in the meanwhile. (Art. 1, No. 66 of 1929.)

The holder of a concession under the decree of May 20, 1924, shall make remuneration (in accordance with regulations to be established by decree) to the holder of a concession right under the decree of September 7, 1882, if on the territory covered by the concession exploitation of coal or petroleum (or both) has been begun or is in progress. (Sec. 3, art. 5, No. 27 of 1924.)

The provisions applying to compensation for damage or surety therefor with respect to prospecting permits apply equally to concessions. (Sec. 4, art. 5, No. 27 of 1924.)

The conditions under which a concessionaire shall operate, as well as the taxes to be levied on the product obtained (royalty ?) will be regulated by decree. (Sec. 2, art. 5, No. 27 of 1924.)

Working Requirements

The holder of a prospecting (searching) permit shall begin operations within one year after receiving the permit and shall continue his investigations to the satisfaction of the Governor. The Governor may in exceptional cases authorize a deviation from this rule. (Secs. 3 and 4, art. 4, No. 27 of 1924.) Should the permittee fail to satisfy these requirements, as well as the one concerning the deposit of damages before beginning operations, the Governor, after hearing the Council of the Administration, and upon presenting the reasons for his action, may withdraw the permit. (Sec. 5, art. 4, No. 27 of 1924.) All these stipulations apply equally to a concessionaire. (Art.6, No. 27 of 1924.)

Transfer

A prospecting permit or a concession may be transferred only with the written consent of the Governor. (Sec. 2, art. 3, No. 27 of 1924.)

Forfeiture

Forfeiture of a prospecting permit or of a mining concession shall result in the following instances (sec. 4, art. 3, No. 27 of 1924):

1. When the holder ceases to comply with the provisions concerning nationality or residence.

2. When at the death of the holder the persons entitled to continue operations have not proved within one year their willingness to comply with the conditions concerning nationality and residence.

Differences of opinion concerning compliance with the requirements of the clause concerning forfeiture will be decided by the judge, according to the manner provided in thedecree of April 1, 1903. (Sec. 5, art. 3, No. 27 of 1924.)

Supplementary Legislation

In addition to the provisions of the coal and petroleum act of May 20, 1924, the following regulations also are in force; Stipulations of articles 1, 5, and 7bis (as far as they are applicable as defined by article 21, second clause, and by article 35) of the decree of September 7, 1882 (as modified and supplemented by the decree of August 30, 1916) and the penal provisions contained in division 6 of the first-mentioned decree, in so far as they are applicable. (Art. 7, No. 27 of 1924.)

BAUXITE LEGISLATION

General

Bauxite legislation is comprised in a general decree concerning the exploration and exploitation of bauxite (No. 80 of 1919) and in a bauxite taxation decree (No. 77 of 1919), together with various amending acts, as follows:

No. 80 of 1919 (Nov. 28, 1919[13]), which combines a decree of March 27, 1918 (No. 76 of 1919) with a decree of Nov. 24, 1919 (No. 78 of 1919).

No. 79 of 1920 (July 2, 1920), amending the bauxite decree.

No. 5 of 1925 (Jan. 19, 1925), amending the bauxite decree.

No. 18 of 1926 (Sept. 28, 1925), amending the bauxite decree.

No. 72 of 1929 (Oct. 9, 1929), amending the bauxite decree.

No. 73 of 1929 (Oct. 9, 1929), granting a special concession for the exploitation of bauxite.

No. 77 of 1919 (March 27, 1918[14]), with respect to the taxation of bauxite.

No. 6 of 1925 (Jan. 19, 1925), amending the bauxite taxation decree.

No. 84 of 1919 (Dec. 30, 1919), regulations in connection with bauxite taxation decrees.

No. 45 of 1925 (June 19, 1925), amending the regulations (No. 84 of 1919) in connection with bauxite taxation.

The bauxite decree, No. 80 of 1919, in article 1, specifically provides that no exploration or exploitation of bauxite shall take place under the law of 1882 or that of 1894. However, the general regulations in articles 1, 2, 5, 7, 7bis, 10, 10bis, 12, 13, 14, 15, 16, 17, 19, 20, 21 (with the exception of clause 3), 22, 23, 28, 29, 30, 31, 32, 33, and 34 of the decree of September 7, 1882 (as it reads after being modified by the decree of August 30, 1916, and the decree of July 2, 1920), and the pertinent penal provisions in part 6 of the same law are applicable. (Art . 13, No. 80 of 1919, and art. 1, No. 18 of 1926.)

The Governor has the authority, after consulting the Council of Administration, to reserve directly or indirectly (through the colonial or the home Govornoent) sections of the country for the exploring or exploiting of bauxite, under special conditions. (Sec. 1, art. 3, No. 80 of 1919.)

13 Enforceable Jan. 1, 1920, by a resolution of Nov. 28, 1919 (No. 79 of 1919).
14 Ibid.

The Governor is entitled also, after hearing the Council of Administration, to grant concessions for the exploitation of bauxite under special conditions, which may differ from the provisions of the decree of 1919. (Sec. 2, No. 72 of 1929.) These conditions are to be made through a colonial decree. (Sec. 3, art. 1, No. 72 of 1929.)

The Governor, with the advice of the Council of Administration, has the authority to refuse (under statement of reasons) or to grant (with the conditions that to him seem necessary) a prospecting permit. (Secs. 1 and 6, art. 5, No. 80 of 1919.)

Prospecting Permits

The holder of a prospecting permit has the prior right to obtain a mining concession, under certain conditions. (Sec. 7, art. 5, No. 80 of 1919.)

Application.--An application for a prospecting permit, which is addressed to the Governor, shall be according to article 3 of the law of 1882 (as modified by No. 1 of 1905). All requests reaching the Governor between 8 a.m. and 1 p.m. on the same day are considered as having been received simultaneously. (Sec. 2, art. 4, No. 80 of 1919.) If these applications refer to the same sections or parts of sections, the order of their consideration shall be determined by lot, with the exception that any request for actual exploitation right shall have the preference. (Sec. 3, art. 4, No. 80 of 1919.) The provisions governing the drawing of lots are covered by sections 4 to 7, art. 4, No. 80 of 1919.

Area.--The area covered by a prospecting permit shall not be more than 50,000 hectares nor less than 5,000 hectares. (Sec. 2, art. 5, No. 80 of 1919.) More than one permit may be granted to one person, but the combined areas under the permits shall not exceed 50,000 hectares. (Sec. 3, art. 5, No. 80 of 1919.)

Duration.--A permit is granted for one year and may be, at the discretion of the Governor, renewed for a period not to exceed one year. (Sec. 5, art. 5, No. 80 of 1919.) In the case of territories for which permission to explore or exploit has been previously given, no permit shall be given until six months have elapsed since the expiration of the previous right. (Sec. 4, art. 5, No. 80 of 1919.)

Priority right to a concession.--A prospecting permit gives to its holder priority right to receive a concession, provided he forms a corporation three months before the expiration of the permit (sec. 7, art. 5, No. 80 of 1919), complies with the provisions concerning the setting up of the company's headquarters in the Netherlands or in Surinam, and complies with articles 11 and 3 of the decree of 1882 (as it is modified by the decrees of March 18, 1908, and January 22, 1903). (Sec. 7, art. 5, No. 80 of 1919.)

Transfer.--A permit holder may, with the written consent of the Government, transfer his prospecting rights for the whole or a part of the area, provided the portion either transferred or remaining shall not be less than

5,000 hectares, and provided the transferee shall not become the holder of more than 50,000 hectares. (Sec. 1, 2, and 3, art. 6, No. 80 of 1919.)

In the case of a transfer, the Government shall issue to the transferee a permit for the unexpired term of the original permit. (Sec. 1, art. 6, No. 80 of 1919.)

Applicable also to transfers are articles 26 and 27 of the 1882 law, as those articles read according to No. 1 of 1905. (Sec. 3, art. 6, No. 80 of 1919.)

Mining Concessions

A bauxite concession does not confer the right to exploit other minerals. The Governor may grant permission to persons other than the concession holder to prospect for or to exploit minerals other than bauxite on the conceded territory; and the concessionaire is obliged to admit the holders of such rights to his territory. (Sec. 1, 2, and 3, art. 11, No. 80 of 1919.) The Governor shall inform the bauxite concessionaire of any additional rights with respect to his concession granted after the coming into force of the bauxite act. (Sec. 4, art. 11, No. 80 of 1919.)

Application.--Applications for the right to mine bauxite shall be made in conformity with the provisions of article 11 of the law of 1882, as it reads after modification by the decree of March 18, 1908. The petitioner shall present a receipt for the payment of the amount due the treasury (paid to the colonial receiver and treasurer) in accordance with article 8 of the bauxite law of 1919. (Sec. 1, art. 7, No. 80 of 1919.)

Duration.--No concession may be granted for less than one year or more than 50 years. (Sec. 2, art. 7, No. 80 of 1919.)

Area.--No concession may be granted for an area of more than 125,000 hectares. More than one concession may be given to one company, provided that the total area is not more than 125,000 hectares. No concession may be given for less than 1,000 hectares, unless the configuration of the area makes the application of such a restriction impossible. (Secs. 2, 3, and 4, art. 7, No. 80 of 1919.)

Rules for the limitation or reduction of an area once granted are found in sections 4, 5, 6, and 7 of article 8, No. 80 of 1919.

Production requirements.--Every concessionaire shall produce each year for six years (beginning from the date of the concession) at least 20 metric tons of mineral from each 100 hectares of land or fraction thereof, calculated according to the total area in the concession. After the sixth year, the minimum yearly production shall be 2 tons a hectare. (Secs. 1 and 2, art. 10, No. 5 of 1925.)

A concessionaire failing to obtain the minimum production for any one year is subject to a fine (to be paid to the colonial treasury) equal to the sum that would have been paid for the lacking production, plus a bauxite tax calculated on the same number of tons. When a concessionaire has been obliged to pay such a fine three times or when he has failed to pay his fine within the period ordered, the Governor is entitled (with the sanction of the Council of Administration), under statement of reasons, to withdraw all or part of the concession. (Secs. 4 and 7, art. 10, No. 5 of 1925.)

The Governor, after hearing the Council of Administration, has the authority to give the necessary instructions for the enforcement of these requirements; or under statements of reasons, he may grant exemptions with respect to minimum production and fines. (Secs. 5 and 6, art. 10, No. 5 of 1925.)

Export duty.--The holder of a concession is obliged to pay to the colonial treasury 25 cents a metric ton or fraction thereof upon all bauxite exported, whether it is in a purified condition or not. (Sec. 3, art. 10, No. 5 of 1925.) (See also section entitled "Bauxite Taxation Legislation," p. 32.)

Rent.--Each concessionaire shall pay to the colonial treasury, in advance (sec. 1, art. 8, No. 80 of 1919, as amended by No. 5 of 1925):

Ten cents a hectare for the first year.
Twenty cents a hectare for the second year.
Fifty cents a hectare for each year following.

With respect to a concession granted for more than one year, the rent is due 30 days before the end of each year; and if this payment is not made, the concession expires, except as further rights may be given by virtue of article 28 of the law of 1882, as it reads according to No. 1 of 1905. (Secs. 2 and 3, art. 8, No. 80 of 1919.)

Transfer.--Written consent from the Governor must be obtained before a concession may be transferred. The Governor will issue a new concession for the unexpired portion of the original concession. A transfer shall not be granted to any one that seeks to acquire more than 125,000 hectares. (Secs. 1 and 2, art. 12, No. 80 of 1919.)

Further provisions concerning transfers are found in articles 26 and 27 of the decree of 1882, as they read according to No. 1 of 1905. (Sec. 2, art. 12, No. 80 of 1919.)

Forfeiture.--A concession legally ceases to be valid if the holder fails to comply with the provisions concerning nationality and residence, or upon the death of the holder if the heirs shall not have within a period of one year signified their willingness to comply with the same provisions. (Sec. 4, art. 2, No. 80 of 1919.)

Differences of opinion concerning compliance with these requirements shall be decided by the judge in the manner laid down in the decree of April 1, 1903. (Art. 1, No. 79 of 1920.)

Terms of a Special Bauxite Concession

A decree passed in 1929 gave to the Governor authority (with the sanction of the Council of Administration) to grant to the Surinam Bauxite Co., in view of its promising prospecting operations, a special concession, the stipulations of which might deviate, if necessary, from the provisions of the bauxite law of 1919 and of the bauxite taxation law of 1919. (Art. 1, No. 73 of 1929.)

The specific stipulations that the Governor was entitled to make are summarized in the following paragraphs.

The concession, which shall cover only the area over which the company already has a right, shall be for 60 years from January 1, 1929. In exchange for the longer than normal period the company shall relinquish a right granted to it by a resolution of March 16, 1925, whereby it has the free use of a telegraphic connection between Paramaribo and Moengo. (Secs. d and e (1), art. 1.)

The duties in force at the time of the issuance of the concession (that is, through the bauxite decree and the bauxite taxation decree of 1919) of 0.50 francs a hectare and 0.25 francs and 0.125 francs a ton of bauxite, shall remain fixed during the life of the concession. No new direct monetary encumbrances shall be imposed; any such encumbrances imposed by a general decree with reference to the bauxite industry shall be refunded to the company, in a manner stipulated by the Governor. (Secs. a, b, and c, art. 1.)

The minimum production of bauxite shall be increased to 150,000 tons a year. The regulations concerning this production shall be the same as those of the general bauxite decree, with the exception that a surplus in any of the three years preceding a year in which there is a shortage shall be taken into consideration. (Sec. e (2), art. 1.)

The concessionaire shall embody the sales price of the shipped bauxite in its profit and loss account for not less than the market value, with, however, a minimum of 12.50 francs a ton according to the grade of bauxite as determined by the Governor. (Sec. e (3), art. 1.)

The company shall make a special contribution to the Government for the upkeep of port and rivers. (Sec. e (4), art. 1.)

Bauxite Taxation

General.--A special bauxite taxation law (No. 77 of 1919) was enacted, together with regulations (No. 84 of 1919) in connection therewith (under authority of section 6 of article 10 of the bauxite law and article 2 of the bauxite taxation law). This legislation applies not only to bauxite produced in the colony but to raw (crude) bauxite imported into the colony. (See also section entitled "Export Duties," p. 28.)

The taxes shall be $12\frac{1}{2}$ cents a metric ton, or fraction thereof, on all bauxite (either pure or mixed) produced in Surinam, as well as on all crude bauxite imported into the colony. (Sec. 1, art. 1, No. 77 of 1919; art. 1, No. 6 of 1925.)

The bauxite tax is computed according to the weight of the bauxite in the condition in which it is shipped or in which it is imported. (Sec. 2, art. 1, No. 77 of 1919.)

Penalties or fines.--Provisions concerning penalties or fines for infringements of the provisions of the bauxite decree (No. 77 of 1919) or of the regulations passed in connection therewith (No. 84 of 1919), which may not exceed three months of imprisonment or 2,000 guilders, are found in articles 3, 4, 5, and 6 of No. 77 of 1919.

Transportation license.--The transportation of bauxite in the colony outside of a concession shall be covered by a transportation license or by a document of importation. Such a license is used as evidence of the minimum production required by article 10 of the bauxite decree of 1919. (Arts. 2 and 6, No. 84 of 1919.)

The license, or permit, is taken over, before or during the transportation of bauxite destined for export by a tax official (appointed by the Comptroller of Taxes). This official notes on the license the quantity (according to his opinion) of bauxite to be exported and files the license in the office of the Receiver of Import Duties and Excises, who collects the taxes. (Art. 12, No. 84 of 1919.) The tax must be paid, or acceptable security must be offered, before export is allowed. If security is given, the tax must be paid with eight days of export. (Secs. 1 and 2, art. 11, and sec. 3, art. 12, No. 84 of 1919.)

Miscellaneous provisions.--Other matters covered by the bauxite decree (No. 84 of 1919) are notifications (to the administrator of domain) by operators with respect to the time of beginning work or the time of discontinuing it, with reasons therefor (secs. 1 and 2, art. 1); notifications to the Comptroller of Revenue of intent to transport bauxite (sec. 1, art. 7); daily records and yearly and quarterly reports by the operator (arts. 3, 4, 5); data with respect to personnel (art. 16); monthly report by the Receiver of Import Duties and Excises to the Administrator of Domain with respect to bauxite imported and exported and its origin and destination (art. 14); rules concerning storage places and storage registers (arts. 8, 9, and 13).

Supplementary legislation.--There shall govern with respect to bauxite taxation, in addition to the taxation decree and regulations in connection therewith, the Navigation Decree, the Tariff Decree of 1922, and the decree of December 5, 1908, regarding the regulations of expenses in connection with the import and the export of merchandise, in so far as they are applicable to bauxite. (Art. 10, No. 84 of 1919; sec. 1, art. 1, No. 45 of 1925.)

I. C. 6718

May, 1933

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

MINING LAWS OF MADAGASCAR

BY

R. M. Santmyers

I.C. 6718
May, 1933.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING LAWS OF MADAGASCAR[1]

By R. M. Santmyers[2]

PREFATORY NOTE

This paper presents one of a series of digests of foreign mining legislation and court decisions that is being prepared in advance of a general report relative to the rights of American citizens to explore for minerals and own and operate mines in various foreign countries. This interpretation of the mining legislation of Madagascar was prepared from the best information (in French) available in Washington, but is released subject to correction and amplification, if necessary, by the proper American diplomatic and consular officers, to whom it will be referred through the courtesy of the Department of State.

INTRODUCTION

The basic mining law of Madagascar is the decree of 1923, signed at Rambouillet by the President of France on July 19, and published in the Journal Officiel de Madagascar et Dépendances. It has been modified by a number of subsequent laws and regulations, generally of minor importance.

All applications made for mining concessions, together with notices of the date for boundary determinations (art. 48) and all other official notices with reference to the mineral industry are published in the Journal Officiel de Madagascar et Dépendances.

All references in this digest, unless otherwise designated, are to articles of the decree of 1923.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6718."

2 Mineral specialist, rare metals and nonmetals division, U. S. Bureau of Mines.

RIGHTS OF FOREIGNERS

The 1923 decree (art. 8) reaffirmed the requirements of previous decrees with respect to the character of persons, associations, and companies, including article 2, paragraph 1 of the decree of January 8, 1916, which reads as follows: "That societies formed for prospecting and exploitation of mines must conform with the French laws and have their main office either in France or in the French colonies." The 1923 decree further requires that the representative in the colony must be furnished with personal authorization. (Art. 10.)

CLASSIFICATION OF MINERAL SUBSTANCES

Mineral deposits are arbitrarily divided into two classes: Quarries and mines. The intermediate classification of "minières" existing in French law does not appear in the Madagascar law.

1. Quarries include deposits of mineral substances such as building materials and fertilizers and analogous substances, with the exception of nitrates and associated salts and phosphates. Peat bogs are classed as quarries. (Art. 2.) Quarries are considered as belonging to the owner of the soil and are governed accordingly. Their exploitation, however, is made subject to regulation by order of the Governor General with the idea of maintaining the safety of the land and assurance of the security of personal employment.

2. Mines include deposits of mineral substances not classed under quarries. (Art. 3.)

The deposits of concessionable minerals are classed as follows: (Art. 5.)

1. Precious metals and stones.
2. Industrial minerals, such as rock crystal, corundum, mica, rare earths, uranium, bismuth, asbestos, etc.
3. Graphite.
4. Iron.
5. Metallic minerals in general other than iron.
6. Mineral salts, such as rock salt, nitrates, and phosphates.
7. Combustible minerals (coal and lignite).
8. Liquid and gaseous hydrocarbons, bitumens, asphalts, and bituminous schists.

Any dispute that arises with regard to legal classification of a substance or mineral deposit is settled by the Governor General in administrative council, based upon the report of the Chief of the Bureau of Mines (Chef du service des mines). (Art. 5.)

PROSPECTING PERMITS

General.- A prospecting permit is considered as a personal right (art. 4) and confers upon its holder the exclusive right to exploit a mineral deposit. (Art. 16.)

If, for any reason the application for a prospecting permit is rejected, the receipt for the money paid is returned to the applicant and may be used in a second application. However, if the second application is also rejected, the receipt is not returned and the payment is retained by the Colony. (Art. 23.)

Prospecting permits are delivered according to date and hour of registering the application, as provided for in article 22. (Art. 17.)

The prospector may freely dispost of all products susceptible of being granted that are the result of his labor after paying the royalties. (Art. 28.)

Area.- The area of a claim is a square defined by its central point and by the length of its sides, which must extend in true north and south and east and west directions. The sides must be at least 5 kilometers in length. (Art. 19.)

No person, association, or company may obtain directly or indirectly a majority interest in several prospecting permits or concessions of a particular mineral or group of minerals, the total area of which exceeds 25,000 hectares, without first securing permission from the Governor-General. (Art. 110.)

Application.- Applications must be in French or accompanied by a duly certified French translation (art. 12) and must be addressed to the Bureau of Mines (art. 11). They must contain the name, surname, profession, nationality, and domicile of the applicant or his agent, if one has been specially authorized; and in the case of a company, the address of the home office, its designation, as well as the designation and domicile of its representative in the Colony. (Art. 21.)

The application must show the size of the claim and the type of mineral or minerals to be exploited. (Art. 21.)

For registering the application, a fixed tax of 150 francs is charged, payable during a period of 12 months preceding the application (art. 18); the receipt must be attached to the application. (Art. 21.)

If the claim lies in two adjoining districts, application must be made to Chief of the Bureau of Mines of each district. (Art. 20.)

Separate application must be made for each claim and for each mineral or group of minerals requested (art. 21).

There may be instituted for the same area, applications by different persons for prospecting permits or concessions, for any of the groups of minerals enumerated on page 2. (Art. 7.)

If two or more minerals, the prospecting or concession rights of which are held by different persons, occur in such a manner that the exploitation of one can not be carried on without the extraction of the other, the person working the deposit must return the mineral to its rightful owner upon receiving payment for extraction costs or else pay a just indemnity. (Art. 7.)

Duration and renewal.- A prospecting permit is issued for a period of two years from date of delivery and may be renewed twice (art. 25). The first renewal is for a period of two years, for which a flat fee of 300 francs is charged and the second is for four years, subject to the annual payment of 400 francs, or 200 francs semiannually. If the mine is producing, these payments may be charged against royalties. (See p. 6, Taxes and royalties.) (Art. 30.)

The request accompanied by the receipt of the previous payment must be sent to the Chief of the Bureau of Mines at least one month before the expiration of the permit. (Art. 31.)

If a part of the claim has been relinquished, the request for renewal must be accompanied by a map showing the land relinquished and the area of the new claim. (Art. 32.)

Transfer.- The renewal of a prospecting permit is considered as a personal right (art. 31), and hence the permit may be ceded or transferred for the whole of an area, though not for a part thereof. (Art. 33.) A tax of 150 francs is paid in addition to any other fees incurred in the transfer. (Art. 33.) If the transfer consists of a voluntary bankruptcy, the declaration must be signed by both the assignor and the assignee and be attested to by the Chief of the Bureau of Mines. (Art. 33.)

Abandonment.- A prospector who wishes to renounce his claim must request permission from the Bureau of Mines. (Art. 36.) A prospecting permit is automatically annulled unless a renewal or concession is requested. (Art. 34.) According to article 35, a prospecting permit is forfeited if (1) the holder is one month behind in sending in his production figures or has not paid up his taxes; and (2) if he has been condemned under articles 85 and 86 (which see).

The holder of a permit which has been forfeited can not again make application for the same claim until after the lapse of four months. (Art. 37.)

CONCESSION PERMITS

General.- A concession confers upon its holder or holders the exclusive right to exploit a given area for a specified group of minerals. It can be issued only to the holder of an unexpired prospecting permit covering the same area. (Art. 40.)

A concession constitutes a real right of limited duration, distinct from ownership of the soil. It is assignable and transferable under conditions of the present decree, and with certain exceptions is subject to the control of the landowner. (Art. 4.)

Area.- The area of a concession must be rectangular; the sides must run true north and south and east and west; and the short side must not be less than one-fourth the length of the long side. The entire area of a concession must fall within the area of the prospecting claim. It shall not consist of less than 100 hectares. (Art. 42.) The same maximum limitation of 25,000 hectares in the aggregate under unified ownership is established for concessions as for prospecting permits unless special permission is granted by the Governor General. (Art. 110.)

Application.- The application for a concession must contain the same information as that required for a prospecting permit with respect to the identity, address, and nationality of the applicant (art. 43), and it must be accompanied by a receipt of payment of a fixed tax of 150 francs, which must have been paid into the Treasury of the colony any time during the 12 months preceding the request (art. 41), and by plans showing the center and all fixed marks of identification. The application must be presented in duplicate (art. 43).

All expenses in connection with the application must be borne by the applicant. (Art. 45.)

Duration.- The duration of a concession is 75 years and may be extended for 25 years more if the concessionaire can prove that sufficient work has been carried on during the period. (Art. 54.)

The application for renewal must be addressed to the Governor General, at least five years prior to the date of expiration. If no reply is received from the application three years prior to expiration, the permit is automatically renewed. (Art. 54.)

Transfer.- The transfer of a concession by sale, donation, legacy, etc., can only be made for the entire area (but see next paragraph, art. 62). The tax collected on such transfers shall not be less than 150 francs. (Art. 61.)

However, under conditions as determined by the Governor General, a concession may be divided, and is then considered as a new concession. (Art. 62.)

Renunciation.- Upon proper application to the Chief of the Bureau of Mines, the concessionaire may renounce his claim under certain conditions. The application for renunciation must be accompanied by a certificate stating that it has been entered in the register and that the taxes have been paid in full. The Governor General may accept the application or place it in adjudication. (Art. 63.)

Forfeiture and nullification.- According to article 64 a permit is forfeited -

1. If production figures are not sent in within one month after notification.
2. If taxes are in arrears more than one month after notification.
3. If claim is divided or sold without authorization.
4. If exploitation ceases without valid reasons.

Forfeiture proceedings may be halted by the concessionaire merely by complying with the regulations which caused the adjudication (see art. 64), paying a fine of 5 francs a day from date of forfeiture, and reimbursing the Government for all expenses incurred. (Art. 65.)

A concession must be placed in operation not later than three years after the granting of the permit. If it is forfeited, the new holder must begin exploitation not later than one year after the granting of the renewal. (Art. 107.)

Nullification extinguishes all rights in the concession. The concessionaire can not reclaim the concession in any manner until after the lapse of one year. (Art. 67.)

If the concession expires or is forfeited or annulled, the concessionaire retains all rights to buildings and constructions within the area of the claim, but those placed upon public domain become the property of the Colony. (Art. 68.)

Taxes and royalties.- The holder of a concession must pay a fixed tax of 50 centimes per hectare per year. It is payable by semester and in advance. (Art. 57.)

A proportional tax (royalty) of 5 per cent of the value of the minerals extracted must also be paid. This tax likewise is payable semiannually. The minimum of the sum of these two taxes shall not be less than 500 francs annually. (Art. 58.)

The rate of the proportional tax is reduced to 2.5 per cent for the first five years from date of concession if the metallic minerals (groups 4 and 5) are treated metallurgically in the Colony. (Art. 59.)

The base for taxation is determined each year by decrees of the Governor General after notice from a consulting committee of the Bureau of Mines. (Art. 58.)

PETROLEUM (AND RELATED SUBSTANCES) REGULATIONS

The regulations of the present decree are applicable to group 8 (gaseous and liquid hydrocarbons, bitumens, asphalts and bituminous schists) with the following exceptions (art. 97):

1. The duration of a prospecting permit is for four years (art. 98) and the fixed tax therefor is 600 francs. It may be renewed twice, but the second renewal as well as the first is for two years; the fee for each renewal is 800 francs. (Art. 99.)

2. The concession is granted for only 40 years, but is renewable under much the same conditions as concessions for other minerals. (Art. 102.)

Renewal of group 8 prospecting permits can be demanded by the holder of a permit provided he has performed certain minimum amounts of actual drilling. (Art. 100.) For the first renewal the requirement is at least 50 meters and for the second renewal, at least 200 meters in total length of holes must have been drilled during the last two years of the life of the first renewal. It is further stipulated that the length of holes less than 50 meters deep is not to be included in the calculation.

The length is accounted for as follows:

1. Less than 100 meters is accounted for at its actual length.
2. Each meter between 100 and 200 meters counts as 2 meters.
3. Each meter between 200 and 300 meters counts as 3 meters.
4. Each meter between 300 and 400 meters counts as 4 meters.
5. Each meter above 400 meters counts as 6 meters.

A holder of several permits may group all or part of the claims when accounting for the assessment work done. If such a grouping is made, the holder may multiply by 50 the amount of work done on all of the permits which have not been renewed, and by 100 the amount of work done on the claim, each year, which have been renewed once or twice, accounting only for the whole years from the date of the first renewal. (Art. 100.)

The tax rate on a group 8 concession is 2.50 francs per hectare per year, and is payable by semester and in advance. It is combined with the proportional tax (royalty); the sum of these two shall not be less than 1000 francs. (Art. 103.)

The proportional tax is reduced to 2.5 per cent during the first five years of the concession, based upon the first 10 permits which can produce 5,000 tons of hydrocarbons annually. (Art. 104.)

EASEMENTS AND DAMAGES

The prospector or exploiter may not occupy any ground within enclosed walls, yards, or gardens without the consent of the owner of the soil. All shafts, pits, or adits must be at least 50 meters from any habitation unless the consent of the occupant is obtained. (Art. 69.) Except by special permission, all prospecting and exploitation is prohibited within 50 meters of all public establishments, such as civil or military prisons, canals, haulage ways, works of art, houses, and tombs. Prospectors and exploiters must also observe all local laws and customs. (Art. 70.)

Article 71 sets forth the right to use the territory within the confines of the claim for the construction of buildings for mechanical treatment of ores, for trenches, canals, and roadways. It also sets forth the conditions relative to the use of water and timber.

Article 72 sets forth the rights to the use of land other than within the perimeter of the claim and adjacent territory.

Article 73 enumerates the conditions relative to the use of paths, roadways, and canals constructed within the vicinity of the claim. Article 74 states that the prospector or the exploiter is held responsible for all damage done to property during the exploitation of the claim. Article 75 sets forth the regulations with respect to damage done to other mines by the flow of water or by other means during the working of the claim.

Each of these articles states that where damage is done an indemnity must be paid either as agreed upon between the prospector or exploiter and the owner of the surface rights or as assessed by court procedure.

FINES

A fine of 1,000 to 25,000 francs or imprisonment from three months to three years or both is imposed upon every person (1) who exploits deposits of precious metals and precious stones without right or who deals in these products without license; and (2) who detains, buys, sells, or put into circulation any of these products. (Art. 85.)

A fine of 100 to 1,000 francs or imprisonment from 15 days to two years or both is imposed upon every person (1) who makes a false declaration as to identity in securing a personal authorization; (2) who knowingly furnishes wrong information in obtaining a prospecting permit, and (3) who falsifies as to the title to a prospecting permit or concession. (Art. 86.)

A fine of 100 to 1,000 francs or imprisonment from 1 to 5 days or both is imposed upon every person (1) who exploits in an illegal manner deposits of minerals other than precious metals and stones; (2) who falsifies as to their output; (3) who opposes the visit of government agents; and (4) who does not keep his books in order. (Art. 87.)

MISCELLANEOUS REGULATIONS

The Governor General in administrative council after notice to the Bureau of Mines has the authority to issue rules and regulations necessary for the execution of the present decree. (Art. 116.)

The Minister of the Colonies is charged with the execution of the present decree. (Art. 118.)

The prospecting and exploitation of mines in Madagascar with regard to the public safety, the safety and health of the workmen, conservation of the mine, the best utilization possible of the deposits, and the conservation of water and public roads are subject to the control of the administration. (Art. 76.) The control of the administration is exercised under the authority of the Governor General by the Chief of the Bureau of Mines and his agents. (Art. 76.)

The prospector and the exploiter must grant to all officials and agents of the Bureau of Mines free access to their properties (art. 81), and they must place before the agents, if requested, plans of the mine workings and the registers that they are required to maintain with respect to amount of ore mined and labor employed (art. 79).

The prospector or exploiter must furnish the Bureau of Mines each year with a copy of the plans showing underground work done during the year, together with all other information relative to output and labor. (Art. 79.)

Claimants of mining concession must defray the expenses of the Bureau of Mines and its agents in establishing claims, marking boundaries, and similar services. (Arts. 33, 45, 56, and 66.)

Civilian employees of the State or Colony and all military officers and soldiers, either on active service, vacation, or on the unattached list are forbidden to take direct interest in prospecting or exploiting mineral deposits in the Colony. (Art. 9.)

I. C. 6719

May, 1933

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

MINING LAWS OF BULGARIA

BY

R. M. SANTMYERS

I.C. 6719
May, 1933.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING LAWS OF BULGARIA[1]

By R. M. Santmyers[2]

PREFATORY NOTE

This paper presents one of a series of digests of foreign mining legislation that is being prepared in advance of a general report relative to the rights of American citizens to explore for minerals and to own and operate mines in various foreign countries. This interpretation of the mining legislation of Bulgaria was prepared from a copy (in French) of the Mining Laws of Bulgaria, promulgated by Royal Decree No. 33 of March 31, 1910, and published in the "Journal Officiel," No. 83, April 14, 1910, and checked against a digest prepared by Maynard B. Barnes, First Secretary, American Legation, Sofia, Bulgaria, dated June 10, 1932, which was transmitted through the courtesy of the Department of State.

CLASSIFICATION OF MINERAL SUBSTANCES

Mineral deposits are classed either as mines or quarries. (Art. 1.)

Mines are deposits, the products from which must be processed. Under this class specific mention is made of the metals (gold, silver, copper, lead, zinc, iron, chrome, titanium, manganese, tungsten, tin, mercury, platinum, cobalt, nickel, cadmium, aluminum, molybdenum, bismith, uranium, antimony, arsenic), sulphur and other "analogous" mineral substances, phosphate rock, iron pyrites, sulphates, saltpeter, alum, boracic acid and its compounds, talc, asbestos, amber, graphite, meerschaum and precious stones of all kinds, rock salt and other salt, peat, coal, petroleum, ozocerite, asphalt and all other bituminous substances. (Art. 2.)

Quarries are deposits that yield mineral colors, chalk, magnesite, gypsum, celestite, barite, fluorite, mica, quartz, feldspar, building and ornamental stones, millstones, lithographic stones, slate, limestone, flint, sand, and all clays. (Art. 3.)

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6719."

2 Mineral specialist, rare metals and nonmetals division, U. S. Bureau of Mines.

Minerals included under "mines" are subdivided into three groups, as follows (art. 14):

1. Peat, coal, petroleum, ozocerite, asphalt, and other bituminous substances.
2. Rock salt, and other salt, including salt springs, etc.
3. All other minerals included under mines but not included in groups 1 and 2.

Any dispute arising with regard to the legal classification is settled by royal decree promulgated at the instance of the Ministry of Commerce, Industry and Labor, after consultation with the Bureau of Mines. (Art. 4.)

The washing of gold from stream beds is not considered as prospecting or exploitation of a mine. Such activity is authorized by a special permit issued, upon payment of 5 leva[3] per person so engaged, by the Prefect of the Department in which the activity is undertaken. (Art. 6.)

OWNERSHIP

Mineral substances classed under mines are held to be of material economic importance and therefore the proprietor of the surface soil has not the right to dispose of them. (Art. 5.)

RIGHTS OF FOREIGNERS

American citizens are permitted to explore and operate mines on the same terms as Bulgarians. According to the law "all physical or juridical persons may obtain the right to exploration and exploitation of mines." Incorporation under Bulgarian law is not necessary. (Art. 8.)

PROSPECTING

Permits

General.- Prospecting permits are issued for a period of two years and may cover either a specific mineral or all minerals falling within any one of the three categories. (Art. 12.) The law contains no specific provision for the renewal of permits but it is understood that in practice renewals are obtainable under conditions approved by the Ministry. It will be noted (see p. 6) that article 27 of the law (as amended) provides for taxes for succeeding periods. Prospecting and the exploitation of a mine are not considered as commerce and therefore are not subject to commercial licenses. (Art. 7.) Prospecting permits are required by law and may be secured from the Ministry of Commerce, Industry and Labor. (Art. 11.)

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3 The lev (plural leva) has been stabilized at 92 leva to 1 gram of fine gold and hence is worth approximately $0.0072 in United States currency.

Application.- The application is made through the Prefecture of the Department in which the property is located and must show the name, surname, profession, nationality, and domicile of the applicant. (Art. 13.)

It must be accompanied by the following documents: (a) birth certificate, (b) certificate attesting that the applicant has not been deprived of his civil rights, wholly or in part, (c) present a guarantee, signed by two responsible persons, before a notary, for payment of all damages that may be caused by him during exploitation, and (d) a receipt covering payment to the Treasury of the area tax for the first year (see Taxes and fees). (Art. 19.)

Within 30 days of applying for a prospecting permit a topographical sketch of a scale not less than 1:126,000 must be deposited at the Ministry. (Art. 20.)

Area.- The claim must be in the form of a rectangle having a minimum area of 50 hectares and a maximum area of 1000 hectares. (Art. 17.)

Priority of right.- Applications for permits relating to the same area are accepted in accordance with the time of the filing of the application. (Art. 24.)

Transfer.- The right to prospect in a given area may be transferred to other persons by mutual consent, by public sale, or by inheritance. (Art. 35.)

When the transfer is made by mutual consent, the contracting parties must notify the Ministry of Commerce, Industry and Labor by declaration sworn to before a notary. (Art. 36.)

When the transfer is made by public sale, the purchaser accepts all rights and obligations covering the claim. (Art. 37.)

When the transfer results from inheritance, the heirs must notify the Minister of Commerce, Industry and Labor not later than three months after the decease of the testator, that they desire to assume all rights to the claim and furnish a new guarantee as required by law (see Concessions, application). (Art. 38.)

Disposal of products.- The holder of a prospecting permit may dispose of all minerals extracted during prospecting, after having informed the Ministry of Commerce, Industry and Labor. In this case, the prospector pays the same royalties as under a concession. (Art. 44.)

CONCESSIONS

General.- All exploitation of mineral deposits is on the basis of a concession granted by the State. The concession is of unlimited duration and is issued only to the holder of a prospecting permit covering the same area.

(Art. 46.) An application for a concession to exploit minerals discovered in an area must be made within two years from the issuance of the prospecting permit. (Art. 34.)

A concession will not be granted unless verification on the ground indicates that (a) exploration work has been sufficient to justify the hope of successful exploitation; (b) that the area requested does not include territory covered by other concessions, and that it coincides with the area shown on the application; and (c) that public interest will not suffer. (Art. 51.)

Concessions may be granted to two or more different persons for different groups of minerals for the same area. (Art. 101.) If the minerals can not be exploited separately, the concessionnaire may mine the minerals and upon delivery to the rightful owner receive payment for extraction. (Art. 102.)

Application.- The application for concession must be addressed to the Ministry of Commerce, Industry and Labor and contain the following (art, 47):

(a) Name, surname, nationality, domicile, and profession of the applicant.
(b) The nature of the minerals discovered.
(c) Place of their discovery.
(d) The name of the concession.
(e) The exact location of the claim and its size in hectares.
(f) Dimensions of the area.

Samples of the mineral or minerals, three copies of a topographical plan of the surface area, and a detailed explanation of the prospecting work done must also be submitted.

The applicant must also deposit a sum of money with the Ministry for verification and correction of the plan (see Taxes and fees).

Area.- The area covered by an exploitation permit may range from a minimum of 50 to a maximum of 800 hectares; in form it must be rectangular, and the short side must not be less than one-fourth of the long side. The extent of the underground workings is determined by vertical planes descending from the surface limits. (Art. 65.)

The concession may extend outside the area of the exploitation in the absence of prior claims to contiguous territory. (Art. 49.)

Transfer.- No mine may be divided and sold in parts without the authorization of the Ministry of Commerce, Industry and Labor. The contracting parties to a transfer must notify the Ministry by a declaration signed before a notary. (Art. 70.)

The joining of two or more mines may take place only where they touch each other, or where, in the opinion of the Council of Mines the underground exploitation may thus be carried on to the best advantage. (Art. 71.)

Renunciation.- Concessions may be renounced by written notice of three months to the Ministry of Commerce, Industry and Labor. Surface and underground plans must be attached to the notification (art. 104).

During the three month's interim the creditors of the concessionnaire may present their claims. In the event that such claims remain unsettled at the end of the three month's period the Ministry may put the mine up for sale at public auction, including all machines and equipment, the proceeds from the sale to be used to satisfy the creditors. (Art. 105.)

In the event that the proceeds of such sale are in excess of the total claims of the creditors, the remainder shall be paid into the public treasury. If there are no creditors or if they do not enter their claims during the three month's period, the mine with all its machinery and equipment becomes the property of the State. (Art. 106.)

Revocation or forfeiture.- If the exploitation of a mine ceases, or is carried out contrary to public interests, for example in the matter of the fixation of prices, the Ministry, on the advice of the Mining Council, may submit a complaint to an arbitral board. In the event that the concessionnaire does not adhere to the decision of this board the Ministry may revoke the concession. (Art. 75.)

If the concession is declared forfeited and is sold at public auction, the proceeds are turned over to the concessionaire after all outstanding debts against the mines are paid. (Art. 107.)

All apparatus and means of exploitation constitute an integral part of the mine and are considered as immovable property (arts. 69 and 108). Extracted minerals, animals, and stock materials are excepted. (Art. 69.)

EASEMENTS AND PRIVILEGES

The concessionnaire has the right, within the limits of the concession, to occupy all grounds necessary for the exploitation of the mine. (Art. 89.) If the ground belongs to the State, he may occupy it after receiving authorization from the Ministry; if it belongs to an individual or to the community, authorization and complete agreement as to indemnities are necessary. (Art. 90.)

If an agreement can not be reached, the Ministry appoints, at the expense of the concessionnaire, a committee which has the right to fix indemnities. (Art. 91.) This indemnity is based upon the revenue received from the land by the landowner, prior to the concessionnaire's occupancy. (Art. 93.)

If in normal course of exploiting the mines, the concessionnaire removes quarry materials, he may use as much thereof as he has need for, but the remainder belongs to the proprietor of the surface land if he removes it within one month and pays for the cost of extraction. (Art. 97.)

Upon payment of an indemnity, a concessionnaire has the right to use roads, bridges, canals, etc., belonging to an adjoining mine, unless it interferes with the working of that mine. (Art. 100.)

If within the limits of a concession or within its immediate vicinity are found State forests, the Ministry may set aside a part of it for use by the concessionnaire after notifying the Mines Council and the Forest Council. The concessionnaire must pay a fixed price for the use of such timber. (Art. 76.)

Concessionnaires may enjoy the free use of any available water power that does not belong to a private owner. (Art. 77.)

All machines, rails, and other equipment and material needed for the exploitation of the mine and not produced in Bulgaria may be imported duty free. (Art. 78.)

Reduced railway tariffs are in force for all locally mined products. (Art. 80.)

Concessionnaires may enjoy the free use of all land, either communal or State, within or in the vicinity of the concession for the construction of roadways, canals, railroads, and other means of communication necessary for the successful exploitation of the mine. (Art. 82.)

Mines and mining equipment are exempt from ordinary property taxes. (Art. 83.)

TAXES AND FEES

A holder of a prospecting permit must pay a fixed tax of one lev per hectare or fraction thereof for each year during the first two years that the claim is "reserved." For the third and fourth years the tax is 2 leva per hectare or fraction thereof. For the fifth and sixth years the basis of the tax is 3 leva per hectare or fraction thereof. The basis of the tax continues thus to mount until 5 leva per hectare or fraction thereof has been reached.[4] (Art. 27.)

A holder of an exploitation permit must pay a fixed tax and a proportional tax (royalty). The fixed tax is payable annually and is 15 leva per hectare or fraction thereof on group 1, and 18 leva per hectare or fraction thereof on

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4 Law modifying taxes as published in the "Journal Officiel" of July 28, 1924.

groups 2 and 3. In addition, all concessionnaires must pay a 5 per cent royalty, calculated on the basis of the value of the refined mineral after deducting transport charges and the cost of hand sorting or mechanical concentration.[5] (Art. 85.)

The budget law of 1921-22 also provided for the following fees:

	Leva
1. For a prospecting permit	500
2. For transfer of a prospecting permit per hectare or fraction thereof	2
3. For an exploitation permit, by royal decree ..	5,000
4. For transfer of a concession per hectare or fraction thereof	10

A fee is assessed for the verification of the plan attached to the application for a concession. (Art. 50.)

A fee is assessed for verification as required under article 51.

FINES

A fine of 20 to 100 leva is levied upon every person who prospects for minerals without a permit, who prospects in a prohibited area, or who does not conform with the law for which there is no set penalty. (Art. 117.)

A fine of 40 to 200 leva is levied upon every person who does not notify the Ministry two months prior to the commencement of work, every person who does not leave at least 10 meters of rock between his mine and the neighboring mine, or who does not adhere to police regulations as laid down in article 116.

A fine of 100 to 500 leva is levied upon every person who knowingly removes or replaces boundary marks of a concession, who operates a mine under the supervision of a person other than a recognized mining engineer or who knowingly falsifies his records so as to pay less royalty. (In the latter case the royalties due must also be paid.) (Art. 119.)

MISCELLANEOUS

No official or employee of the Mines Branch nor the wife of any such person is allowed to prospect or exploit a mine, except in the case of direct succession. (Art. 9.)

The concessionnaire or his agent or representative must have a domicile in the Department in which the mine is located. (Art. 72.)

5 Budget Law of 1921-22, modifying taxes and fees.

Exploitation must be conducted under the management of an engineer having the right of free practice in Bulgaria. (Art. 74.)

Concessionnaires are liable both before the civil and criminal courts for all acts resulting from negligence or contravention of the exploitation regulations. (Art. 121.)

No underground work shall be carried further than to within 10 meters of the vertical boundary of the concession, except by authorization from the Mines Service. (Art. 98.)

Adjoining concessions must extend aid to each other in all mine-rescue work. (Art. 99.)

No shafts or galleries may be sunk within 75 meters of an abode, without the authorization of the landowner. In special cases this distance may be lessened by authority of the Ministry after notice from the Mines Council. (Art. 88.)

I. C. 6720

May 1933
(Revised May 1935)
(Second Revision December 1936)

Department of the Interior

UNITED STATES BUREAU OF MINES
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INFORMATION CIRCULAR

VERMICULITE

BY

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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

VERMICULITE[1]

By Alice V. Petar[2]

FOREWORD

Although known to mineralogists for many years, vermiculite was of little or no commercial value prior to 1925. It was described in the American Journal of Science in 1824 by T. H. Webb, who named it vermiculite from the Latin vermiculari, to breed worms, because of its property of expanding and unfolding into worm-like forms when heated. It is said that in Japan it was a popular amusement to throw vermiculite on hot coals to see it exfoliate.

The commercial development of vermiculite has been largely due to the efforts of one company, The Zonolite Co., of Libby, Montana, which controls large deposits of raw material and has developed diversified markets for its product. Vermiculite is already an important ingredient of many insulating materials and other articles of commerce, and interest in the material among both producers and consumers is evidently growing.

DESCRIPTION AND PROPERTIES

The term "vermiculite" is applied to a group of micaceous minerals that generally are alteration products of biotite, phlogopite, and other varieties of mica. The original cleavage is partly retained; other physical properties and the chemical composition show varying degrees of alteration. The most pronounced characteristic of vermiculites is their extraordinary expansion on heating; the volume may increase up to 16 times the original. Some varieties contain as much as 20 per cent of water and the expansion occurs as the water is given off. This expansion or exfoliation takes place in only one direction, at right angles to the cleavage. At the same time the color changes from black or dark brown to a silvery or golden hue, according to the degree of heat and the exposure to the air. This change of color is believed to be due to the oxidation of the iron, and it may be controlled to a limited extent by excluding air during heating. In an atmosphere of diminished oxygen content the oxidation is incomplete and the color decreased in intensity. The specific

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1 The Bureau of Mines will welcome reprinting of this article, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6720."

2 Rare metals and nonmetals division, U. S. Bureau of Mines.

gravity of the raw material is in the neighborhood of 2.5, whereas after heating, the expanded mass has been reported to have an apparent gravity as low as 0.087. The untreated material weighs about 100 pounds per cubic foot; after heat-treatment the average product varies in weight from 6 to 20 pounds per cubic foot.

Under the heading "Vermiculites" Dana lists a number of minerals, the best known of which are vermiculite and jefferisite. Other varieties are kerrite, lucasite, lennilite, hallite, painterite, philadelphite, protovermiculite, vaalite, maconite, dudleyite, pyrosclerite and roseite. These minerals show great similarity and in most instances the names are derived from the localities in which specimens have been found.

USES

Largely as a result of research and sales promotion on the part of the principal producer of vermiculite, many uses have been developed. Vermiculite is of little or no value in its raw state, and the following applications relate to the treated material. Its value in heat and cold insulation has been demonstrated and it is an effective sound insulator. Vermiculite has been recommended as an insulating material in fireless cookers, incubators, ovens, pipe and boiler coverings, and refrigerators. When used to fill hollow spaces over ceilings it is said to make houses warmer in winter and cooler in summer. It has been used as an insulator in safes and filing cabinets with apparent success. As a sound deadener it is of particular value in moving picture studios and apartment houses. Mixed in a plaster with wood pulp, vermiculite is said to have been tried out on the walls of Canadian theaters for its acoustic properties with satisfactory results. It is, of course, fireproof.

Vermiculate is used as a fine aggregate in place of sand in connection with gypsum plaster for interior plastering purposes. The cost of such walls is said to be about 5 cents per sq. yd. greater than when ordinary sand is used, but the weight is considerably less and the walls have good insulation properties.[3] In experimental work performed at the University of Kentucky, Lexington, C. S. Crouse[4] reports that a plaster composed of 30% plaster of Paris, 60% calcined vermiculite, and 10% asbestos was not disintegrated when placed in a red-hot furnace. Fireproof insulating board made from vermiculite is said to stand exposure to 1700° F. without any appreciable expansion or contraction.[5]

The golden color of treated vermiculite is utilized in making gold paint. The material may also be used as a paint pigment or kalsomine by tinting to the desired color. Vermiculite is also used as a decorative material in wallpaper.

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3 Rock Products. Progress in the Use of Zonolite. Vol. 35, April 23, 1932, pp. 17-18.
4 Crouse, C. S. Calcined vermiculite as a plaster base. Eng. and Min. Jour., vol. 128, No. 24, Dec. 14, 1929, pp. 923-924.
5 Rock Products. Colorado Vermiculite - Its Discovery and Development. Vol. 35, Aug. 13, 1932, pp. 22-24.

With asphalt binders or tar adhesives vermiculite has been used as a composition roofing. A product marketed as Zonoasphalt is described as a fire-proof roofing material, guaranteed for 20 years without painting or repairing.

The lubricating qualities of vermiculite are said to be comparable to those of flake graphite. It has the property of coagulating or hardening oils so that it may be used instead of aluminum stearate, and at the same time serve as a valuable lubricant. When so used the bearing surfaces become coated with the soft flaky material and friction is reduced to a minimum. For this use the vermiculite should be water ground to about minus 300 mesh.[6]

Insulating cements made from Colorado vermiculite are on the market. They are claimed to have a covering capacity as high as 60 sq. ft. per 100 lb. wet and no volume or linear shrinkage when dried; the dry weight per cu. ft. in place will not exceed 20 lb.

Vermiculite is also used in an insulating brick which is said to have a crushing strength of 175 - 200 lb. per sq. in. and a density of 22.5 lb. per cu. ft. The brick will stand constant exposure of 1850 deg. F. without appreciable shrinking or checking.[7]

Other products in which vermiculite is used to advantage are automobile mufflers and high temperature gaskets.

MODE OF OCCURRENCE[8]

Vermiculite is a constituent of some altered igneous rocks and it generally occurs mixed with other minerals and thus distributed through the rock mass. In a few places, however, it is found in almost clean or unmixed bodies large enough to mine. Bodies that have a dikelike form and range from 1 to 6 feet in width are reported from North Carolina.

DOMESTIC DEPOSITS

Montana

The largest known deposits of vermiculite are in the lower part of the basin of Rainy Creek, about 7 miles northeast of Libby, Montana. The area is

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6 Rock Products. Progress in the Use of Zonolite. Vol. 35, April 23, 1932. pp. 17-18.

7 Rock Products. Colorado Vermiculite -- Its Discovery and Development. Vol. 35, Aug. 13, 1932, pp. 22-24.

8 Pardee, J. T. and Larsen, E. S. Deposits of Vermiculite and Other Minerals in the Rainy Creek District near Libby, Montana. U. S. Geol. Surv. Bull. 805-B, 1929, pp. 22-23.

accessible to the main automobile highway along the north bank of the Kootenai River and is within a few miles of the Great Northern Railway. This occurrence is described by Pardee and Larsen,[9] as follows:

"A body of vermiculite in the Rainy Creek district that is being developed by the Zonolite Co. on the spur north of Kearney Creek is much larger than any deposit heretofore known. It presents no natural exposures, but its outcrop and the slopes below are mantled with a yielding slippery soil composed chiefly of mica-like flakes. As incompletely shown by the workings so far made, this body appears to be of dikelike form and at least 100 feet wide, and 1,000 feet long. It extends to a depth of more than 100 feet, its lower limit not being shown. Several smaller bodies of vermiculite occur in the ground of the Vermiculite & Asbestos Co. on the northwest slope of the same spur. There a tunnel penetrates six bodies that range from 1 to 4 feet in width. They are of flat lenslike or tabular form, and most of them are definitely separated from the wall rock by fault or slip planes. Incomplete exposures of several other similar bodies are made by smaller workings. * * * Samples representing areas of several square feet at different places in the workings of the Vermiculite & Asbestos Co. contained from 30 to 84 per cent of vermiculite. Apparently there is a huge amount of such mixed material."

The property now operated by the Zonolite Co. was accidentally discovered in the Spring of 1916 by E. N. Alley, when he was prospecting for other minerals. While examining a quartz stringer in an old tunnel he noted that the heat from the candle which he had stuck in the wall had caused a leaf of mineral to swell. He took some of the material home and on heating it on the stove and in the fire found that it did not burn but swelled to many times its original size and turned a gold color. It resembled gold nuggets but was light as a feather, which caused it to be known for some time as "feather gold."[10] It was soon apparent that there was an abundant supply of material that possessed remarkable properties, but no use was known for it. Mr. Alley glimpsed the commercial possibilities of vermiculite and, after several years of experimentation, joined with other interested parties to organize the Zonolite Co. at Libby, Montana, to exploit the deposits. The company developed a heat-treated vermiculite which it markets under the trade name "Zonolite," and for which it has found many actual and potential uses. Commercial production was undertaken on a small scale in 1925. In 1931, the Dominion Stucco Co. Ltd., of St. Boniface, Manitoba, a subsidiary of Gypsum, Lime and Alabastine, Canada, Ltd., acquired an interest in the company, and has subsequently promoted the sale of zonolite in Canada.

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9 Pardee, J. T. and Larsen, E. S. Deposits of vermiculite and other minerals in the Rainy Creek District near Libby, Montana. U. S. Geol. Surv. Bull. 805-B, 1929, pp. 23-24.

10 Pit and Quarry. Quarrying and Refining Zonolite. May 23, 1928, p. 82.

As freight rates on crude vermiculite are lower than on the expanded material, the Zonolite Co. finds it advantageous to ship the crude material, pre-screened to the proper size and pre-dried of surface moisture, from the mine to points of consumption. The Zonolite Products Co., an affiliated company operates expansion units at Joliet, Illinois and Brooklyn, N. Y., and distributes zonolite throughout the territory east of the Mississippi; the Western territory is handled by the Zonolite Sales Co.

The Vermiculite & Asbestos Co. was organized early in 1927 to exploit deposits near Libby, Montana, and for a time the Micalite Co. operated several claims in the same region.

According to the U. S. Geological Survey, the following occurrences in Montana have not been described in published reports: Sec. 19, T. 28, N., R. 16 E., in the Rocky Boy Indian Reservation, Hill County, owned by the Bearpaw Mining and Milling Co., Havre, Montana, and about 12 miles east of Hamilton, at the head of Gird Creek in the Sapphire Mountains, Ravalli County, owned by S. H. Chamberlain, Victor, Mont.

Colorado

Half a dozen promising occurrences of vermiculite have been reported in Colorado. Alderson[11] refers to three, as follows:

> "The first discovery of jefferisite in commercial quantity was made by W. B. Thomas in 1913, in the Turrett mining district, 14 miles north of Salida, Colo. The deposit occurred as a vein, 20 inches in width, in a granite formation. About 8 carloads were mined and shipped, but on account of the high cost of mining, the venture was unsuccessful. Later another deposit, 9 miles from Iola, Gunnison County, was discovered. * * * W. B. Thomas and others, in October 1923, visited the main Colorado deposit. The jefferisite is found 7 miles from Westcliffe, the county seat of Custer Co., an old and well known mining district. The company has a tract of 80 acres on which the jefferisite is well exposed in seven places. * * * On an adjoining tract of 60 acres, under lease by the Jefferisite Products Co., are 5 openings, 2 shafts 10 and 12 ft. deep, and 3 open cuts, all of which show jefferisite."

The deposits at Turrett and at Westcliffe were operated in 1929.

Vermiculite has been mined by the Denver Mining & Manufacturing Co. from a deposit about 5 miles southeast of Hecla, and the heat-treated product sold under the name "Tung Ash." J. M. Kyrl, of Chicago, operated the Goldenite and Silverite mine in Fremont County, Colo., near Hillside, in 1931, and

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11 Alderson, Victor C. Jefferisite. Colo. School of Mines, Circ. of Inf., undated, 4 pp.

development work was reported at several other vermiculite properties during 1932. Non-Metallics, Inc., started to develop a vermiculite deposit about 9 miles west of Rye, Colo., owned by E. F. Gobatti, of Pueblo,[12] and a deposit at Feldspar was worked by L. D. Christison, who also owns a deposit of lower grade material near Cotopaxi. Colorado vermiculite is treated by the George B. Smith Chemical Works, Inc., of Springfield, Ill., and manufactured into insulating bricks and blocks, sectional pipe covering, insulating cements, etc. It is reported that Gustavus Sessinghaus, of Denver, shipped several tons of vermiculite in 1932 from a deposit 8 miles from Buena Vista, Colo.

North Carolina

The presence of vermiculite in North Carolina has been known for many years, but until recently there has been little or no commercial production. The variety known as Culsageeite was reported upon in 1873 by Cooke,[13] and Ross and Shannon[14] have described an occurrence of nickeliferous vermiculite about one mile east of Webster, N. C. Within recent months there has apparently been a growing interest in the development of vermiculite deposits in the State, and shipments have been made by at least one producer.

Pennsylvania

The variety of vermiculite known as Jefferisite was discovered many years ago in a serpentine quarry near West Chester, Penna., by William W. Jefferis, for whom it was named. The deposit, known locally as the Brinton Quarry, was worked in 1929 by the John Warren Watson Co., of Philadelphia.

Wyoming

A deposit of vermiculite near Encampment, Wyoming, is being developed by the Parco Development Co.; it is reported that the company shipped 44 tons of the material from its property in 1931.

Other States

Occurrences of vermiculite have been encountered elsewhere in the United States, but commercial developments appear to be limited to the above-mentioned localities.

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12 Mining Journal (Arizona) Vol. 16, No. 5, July 30, 1932, p. 20.
13 Cooke, Josiah P., Jr. Culsageeite, the Vermiculite of the Jenks Mine, North Carolina. Am. Acad. Arts and Sci., Proc., vol. 9, 1873, pp. 48-59.
14 Ross, Clarence S. and Shannon, Earl V. Nickeliferous Vermiculite and Serpentine from Webster, North Carolina. Am. Mineralog., vol. 11, No. 4, April, 1926, pp. 90-93.

MINING AND TREATMENT

The methods of mining and treating vermiculite employed by the Zonolite Co. are described as follows:[15]

"At the present time the property is being worked from the top as an open quarry. The overburden, which varies from 10-in. to 2 ft., is scraped off and a face of pure zonolite about 30 ft. wide and 150 ft. long constitutes the present working pit. Engineers estimate that the deposit contains at least 25,000,000 tons of the mineral. * * *

"On account of the transportation problem it soon became apparent that the raw mineral would have to be shipped to industrial centers and expanded near the place of use. In 1925 the Zonolite Co. built a large plant at Libby to expand the mineral and the plant is still in existence but the modern trend is toward shipping the raw rock. This would call for a large number of plants throughout the east which, obviously, for the tonnage available per plant, had its disadvantage. So here again the problem is being met in a novel manner.

"The idea was to develop a small 'expander' which would be leased on a royalty basis. The machine would represent a small investment and its capacity would be in the neighborhood of two tons of zonolite per hour. In working out the design of such a device it was found that heat alone was not all that was necessary to secure a maximum expansion of the mineral. In the rotary type kiln the best expanded material weighed 7 to 9 lb. per cu. ft. on the ½-in. material. On the new type expander, a description of which follows, the mineral has been expanded to weigh as low as 5 lb. per cu. ft. Material of this weight is so light that the direct flame from an oil or gas burner literally blows it out of the furnace. * * *

"In using the gas or electrically heated machine it was found advantageous to size the crude zonolite before expanding it. This is done by using a "Jigger" vibrating screen that gives three sizes of zonolite: ½-in., 1/8 in. and fines. These are expanded separately and almost instantly.

"The furnace consists of a machine which is also known as a "Jigger." A vibrating plate 3 ft. wide and 7 ft. long and set at a slight slope is enclosed in a firebrick housing hung independent of the plate. When gas is used as fuel the heat is applied above the vibrating plate through suitable port holes and where electricity is used the electric heating elements are hung about 5 in. above the plate. The sized raw zonolite is fed to the top of the vibrating

15 Rock Products. Progress in the Use of Zonolite. Vol. 35, No. 8, April 23, 1932, pp. 17-19.

plate and as it descends is thrown up into the heated zone and this results in a uniformly expanded material at low cost. The machine has a capacity of 4000 lb. per hour and requires a 3-hp. motor to operate the vibrating mechanism. The electric heating elements are rated at 45 kw. and use 220 volt, 3-phase alternating current. They will withstand a temperature of 1800 deg. F. The machine and process are protected by patents and it is proposed to lease the machine on a royalty basis."

FREIGHT RATES

Because of its light weight and consequent bulk, expanded vermiculite carries a high freight rate ($65.00 per ton on shipments from Libby, Montana to Eastern points, in less than carload quantities), which makes it impracticable to ship the material between distant points. To meet this situation the Zonolite Co. has adopted the plan outlined on page 7. The company ships the crude ore, pre-screened to the proper size, and pre-dried of surface moisture, from the mine to points of consumption. Freight rates on vermiculite in this form are as follows: To the Chicago district, $10.00 per ton; to Pittsburgh, Cleveland, etc., $12.00; and to the Eastern Seaboard, $15.00 per ton.

The following freight rates, supplied by the Interstate Commerce Commission, governing various forms of mica, apparently apply to vermiculite:

Crude vermiculite 1/ Carloads, min. wt. 80,000 lb. (cents per 100 lb.)		Ground vermiculite 2/ (cents per 100 lb.)		
From:	Libby, Mont.	Libby, Mont.	Denver, Colo.	Asheville, N. C.
To:				
Chicago, Ill.	50	50	75	997 3/
Joliet, Ill.	50	50	75	997 3/
St. Louis, Mo.	60	60	75	900 3/
Boston, Mass.	75	85	--	850 3/
New York, N. Y.	75	85	--	750 3/
Philadelphia, Pa.	75	85	--	630 3/

1/ Classified as "crude hydrated biotite mica."
2/ Classified as "ground mica."
3/ Cents per 2000 pounds.

PRICES[16/]

Little information is available with reference to prices of vermiculite. In September 1932, the Zonolite Products Co. (a unit of F. E. Schundler & Co., Brooklyn, N.Y., and largest distributor of the output of the Zonolite Co.) quoted pre-dried and screened vermiculite at $15.00 per ton, f.o.b. Libby, Montana, in bulk. The expanded material is sold from the company's expanding plant at Joliet, Illinois at $45 per ton on sizes which do not require particular grinding. In March 1933, quotations for vermiculite appeared in Metal and Mineral Markets, the figure being $7 per ton, f.o.b. mines, North Carolina. In June 1934 this was increased to $7.50 per ton.

POSSIBLE PRODUCERS

L. S. Rees, Western Rawlplug, Inc., 1156 California St., Denver, Colo.
Zonolite Co., Libby, Montana.
Bearpaw Mining & Milling Co., Havre, Mont.
Manganiferous Iron Co., 500 Minnesota Bldg., St. Paul, Minn.
J. K. Chalmers, 632, H. W. Hellman Bldg., Los Angeles, Calif.
R. E. Tilden, 2829 Benvenue Ave., Berkeley, Calif.
Allied Minerals, Inc., Pueblo, Colo.
D. E. C. Austin, Turret, Colo.
John Warren Watson Co., West Chester, Penna.
Industrial Minerals Corp. of Amer., 220 Delaware Ave., Buffalo, N.Y.
R. G. Rogers, Boice Hardwood Co., Inc., Hayesville, N.C.
S. A. Jones, Waynesville, North Carolina.
Norman S. Poole, Hayesville, North Carolina.
Roland W. Ainsworth, Hillside, Ariz.
Gustavus Sessinghaus, Engineers Bldg., Denver, Colo.
Arthur Flannigan, 424 N. 3rd St., Canon City, Colo.
Roy M. Gilliam, Box 131, Pony, Montana.
A. H. MacDougall, 218 W. 23rd St., Cheyenne, Wyo.
G. W. Oliver, Box 271, Penrose, Colo.
Earle H. Paine, Parco, Wyo.
National Vermiculite Products Corp., Chicago, Ill.
Mrs. Winona Sparling, Rosita, Colo.
Mrs. Catherine Carrau, Ennis, Montana.
Vermiculite Products Co., Inc., 882 South York St., Denver, Colo.
Glenn Rathburn, Paint Gap, North Carolina.
H. A. Coggins, Swannanoa, North Carolina.
Dr. Charles P. Edwards, Asheville, North Carolina.
Howard N. Butler, Sanford, North Carolina.
R. L. Corbin, Dillard, Georgia.
J. D. Parsons, Gunnison, Colo.
Harford Talc & Quartz Co., 4 Rockford Building, Towson, Md.

16/ Revised May 1935.

POSSIBLE BUYERS[16]

K. L. Conley, Vancouver Plywood Co., 333 N. Michigan Ave., Chicago, Ill.
Johns-Manville Research Laboratories, Materials Development Section I, Manville, N.J.
F. E. Schundler & Co., 45-15 Vernon Blvd., Long Island City, N.Y.
John Wiener, 81 W. 7th St., Oswego, N.Y.
Morgenstern Asbestos Covering Co., 3951 Utah Street, St. Louis, Mo.
George R. Hall & Sons, 11 E. 44th Street, New York, N.Y.
Industrial Minerals Corp. of America, 220 Delaware Ave., Buffalo, N.Y.
Southern Mica Co., Franklin, N.C.
Sydney L. Smith, 500 Fifth Avenue, New York, N.Y.
J. A. Webb, 25 West 75th Street, New York, N.Y.
Kay M. Grier, 906 Vine Street, Los Angèles, Calif.
Farnum & Co., Chicago, Ill.

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Shannon, E. V. Vermiculite from the Bar Hills near Baltimore, Md. Am. Jour. Sci., vol. 15, 1928, pp. 20-24.
Steele, William S. Vermiculite - Production and Marketing by the Zonolite Company. Am. Inst. Min. and Met. Engrs. Trans., vol. 109, 1934, pp. 418-426.

16/ Revised May 1935.

RECENT DEVELOPMENTS

By M. A. Cornthwaite[17]

At present the main outlet for vermiculite is in the domestic field, for house insulation. A second important use which has been developed recently is insulation of open-hearth steel furnaces. Exfoliated vermiculite granules may be raked loosely on the tops of furnaces and it may be covered with a "coating" or cement composed of vermiculite, mineral fiber (asbestos or mineral wool), and bond (e.g. bentonitic clay). This insulating coating also may be plastered on vertical walls and other surfaces. Insulating brick made from vermiculite granules and a ceramic bond has a high insulating value combined with extraordinarily light weight (18 ounces) and a strong, tough structure. Several concerns are marketing "motor-dopes" for use in automobile or other internal-combustion engines. A mixture of raw and expanded vermiculite and lubricating oil, introduced between the surfaces of the piston and cylinder, forms a seal, it is claimed, and thus prevents power leakage in the motor. The Bureau of Mines has made no tests of this procedure, but the use of both raw and exfoliated vermiculite for this purpose is covered by United States Patents 2012951-2012952 issued September 3, 1935 to Harold S. Brinker and Wm. B. Thomas. One of the latest developments is the use of expanded vermiculite in packing fragile articles. Following is a list of uses classified according to size or mesh of expanded material and is based upon a tabulation prepared for the Tennessee Valley Authority.

	1/4-inch to 20-mesh	
House insulation	Safe and vault linings	Smelter ladles
Home refrigerators	Pipe covering	Refractory brick
Auto mufflers	Boiler lagging	Insulation cement
Acoustic plaster		
	20- to 40-mesh	
Auto insulation	Passenger-car insulation	Fire extinguishers
Airplane insulation	Wall board	Filters
Refrigerator car insulation	Water coolers	Cold storage
	Annealing steel	
	40- to 120-mesh	
Linoleum	Cornice boards	Dielectric switch-boards
Shingles		
	120- to 200-mesh	
Grease lubricant	Bakelite products	Tires and rubber goods
	200- to 270-mesh	
Wall paper printing	Building up viscosity in oil	Fireproof cartons for films
Outdoor advertising paints		

270-mesh

Extender for gold and bronze printing ink or for paint.

17/ Nonmetal Economics Division, U. S. Bureau of Mines.

Prices of vermiculite vary according to the locality, standard-grade raw material, suitable (after being expanded) for house insulation purposes, ranging from $12 to $20 a ton, the average probably being between $14 and $16 per ton in wholesale quantities. North Carolina raw vermiculite is quoted in trade journals at $7.50 per ton, f.o.b. mines. In Omaha expanded material is offered at 14 cents per cubic foot in bulk or 15.5 cents in bags, the bulk price corresponding to around $46.60 per ton. In Washington, D. C., single bags containing enough material to cover 18 square feet 3 inches deep have been offered at 99 cents per bag delivered, equivalent to $74.50 per ton.

The most extensive deposits of vermiculite occur in Montana about 7 miles from Libby in a mineralized zone about 2 miles long and about 1800 feet wide. The east end of this deposit is owned by the Zonolite Corporation and the west end by the Universal Insulation Co. The former concern was founded by Edgar M. Alley, but is now controlled by Fisher Bros., Detroit, and William B. Mayo and associates. In addition to operating the mine and mill at Libby, this company has sales agreements with the F. E. Schundler Co. (plants at Joliet, Ill., and Long Island City, N. Y.), Zonolite Insulation Co. (plants at St. Louis, Tulsa, Kansas City, and Denver), and other concerns. A total of about 25 plants is planned, each to be situated at a large consuming center, for expanding vermiculite to avoid excessive freight due to the bulky nature of exfoliated material.

The Universal Insulation Co. succeeds the National Vermiculite Products Corporation of Chicago, which in 1934 acquired the property and assets of the Vermiculite and Asbestos Co., Libby, Mont. This concern has built several expanding plants in the East and has erected a new mill at Libby for cleaning 75 tons a day of vermiculite. The Mikolite Co., Kansas City, Mo., not only produces house fill but also a full line of insulating plaster, acoustical plaster, roof insulation, and decorative finishes. The raw material is from Wyoming and is said to differ from other vermiculites by expanding like a sponge instead of like an accordion.

The Associated Minerals, Inc., (Ralph J. Hole, president) supersedes the Allied Minerals Co., (H. O. Aaberg, president). The new company, after abandoning several properties in Colorado, Wyoming and Montana has concentrated its efforts on a new deposit at Gunnison, Colo. Shipments have been made from several localities in Colorado, namely Hillside, Boneyard Park, Dead Mule Gulch and Salida. The Sparling mine at Boneyard Park has been leased to Guthrie Cole and E. H. Wheelright, who have built a small exfoliating plant at Canon City.

A number of companies and individuals were mining vermiculite or doing development work in North Carolina in 1935, production being reported by Philip S. Hoyt, Franklin, N. C., and others. Shipments were also made by the Mikolite Co. from Encampment, Wyo.

Figures showing total production of vermiculite are not available, but consumption in the United States, almost exclusively from domestic sources except for some experiments with Russian material, doubtless approached 15,000 tons in 1935.

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ACCIDENT EXPERIENCE OF FOUR LOUISIANA PETROLEUM REFINERIES

BY

F. E. Cash

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

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By F. E. Cash[2]

Among petroleum refinery employees and officials throughout the United States there has been marked effort to reduce accident frequency and severity. For the interest and benefit of refineries both in Louisiana and in other petroleum-refining States it has been possible to obtain from five refineries, representing 70 per cent of the production of Louisiana, the accident experience for the past 3 to 6 years.

The companies concerned have requested that their names be not used. For this reason the four plants listed and discussed are designated as refineries A, B, C, and D. The size of the plant as regards largest to smallest number of employees is indicated by the alphabetical order.

Comparative Accident Experience

According to The Safe Worker for July 1932, published by the National Safety Council, petroleum ranked fourteenth in frequency and sixteenth in severity among 28 industries in 1931.

Table 1 gives the industrial injury rates, arranged in order of severity.

As applied to this table and throughout this discussion, the following definitions are used:

Frequency is the number of lost-time injuries per million man-hours of exposure.

Severity is the number of days lost as a result of these injuries per thousand man-hours exposure.

Severity or days lost per thousand man-hours of exposure as a result of injuries appears to be the most nearly logical basis for comparison of accident experience, although many if not most safety experts would probably not concur in this conclusion.

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6721."

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Table 1.- Industrial injury rates in 1931, according to the National Safety Council

Industry	Frequency		Severity	
	Rate	Rank	Rate	Rank
Average industries	15.12	-	1.72	-
Printing and publishing	9.12	4	0.25	1
Glass	11.31	9	.54	2
Tanning and leather	13.73	13	.56	3
Textile	9.11	3	.58	4
Machinery	9.57	6	.90	5
Automobile	9.48	5	.94	6
Meat packing	29.13	24	.99	7
Food	15.86	17	1.01	8
Rubber	11.78	10	1.03	9
Metal products	14.70	15	1.17	10
Nonferrous metallurgical	9.88	7	1.18	11
Electric railway	19.73	18	1.46	12
Paper and pulp	20.62	19	1.52	13
Chemical	12.65	11	1.84	14
Refrigeration	32.18	25	2.00	15
Petroleum	14.14	14	2.06	16
Public utilities	12.76	12	2.08	17.5
Railway car and equipment	15.48	16	2.08	17.5
Foundry	24.19	22	2.10	19
Steel	10.87	8	2.22	20
Woodwork and lumbering	33.54	26	2.60	21
Laundry	6.24	2	2.75	22
Cement	4.86	1	2.80	23
Marine	24.28	23	2.87	24
Ceramic	22.89	20	3.21	25
Construction	48.15	27	5.14	26
Quarry	22.98	21	6.88	27
Mining	57.24	28	9.44	28

Louisiana Petroleum Refineries

According to United States Bureau of Mines' Information Circular 6641, Petroleum Refineries in the United States, January, 1932, published in July, 1932, there were 473 petroleum refineries in the United States; 14 of these with a combined daily capacity of 224,400 barrels of petroleum were in Louisiana. The five refineries in Louisiana, all different types, from which accident information was obtained for this publication, have a daily capacity of 157,000 barrels and each is a different production and distribution unit. Since the four refineries A, B, C, and D, have more than 70 percent of the refinery capacity of Louisiana, they should give representative data for the State.

Table 2 gives the accident frequency and severity rates, with the essential data to obtain such figures, for the four plants, A, B, C, and D. As contrasted with plants A and D, it will be noted that plants B and C show an annual improvement in both accident frequency and severity.

Table 2.- Accident frequency and severity rates at refineries A, B, C, and D

Year	Average number of men employed	Exposure, man-hours	Injuries: Fatal	Injuries: Partial permanent	Injuries: Temporary	Injuries: Total	Total days lost	Frequency rate	Severity rate
				PLANT A					
1931	3,600	8,674,915	2	-	29	31	15,435	3.57	1.78
1930	4,556	12,268,666	1	3	69	73	11,152	5.95	.91
1929	4,601	12,546,864	2	1	99	102	17,577	8.13	1.40
1928	4,487	12,076,688	5	4	124	133	36,914	11.01	3.06
1927	4,700	12,223,092	7	2	95	104	47,704	8.51	3.90
1926	5,300	12,846,760	2	5	162	169	20,332	13.16	1.58
				PLANT B					
1931	502	1,165,293	-	-	28	28	359	24.03	0.31
1930	644	1,317,527	-	-	77	77	455	58.44	.35
1929	578	1,404,861	-	1	108	109	583	77.59	.41
1928	616	1,398,361	-	-	159	159	841	113.70	.60
				PLANT C					
1931	1,256	3,828,090	-	-	1	1	10	0.26	0.0026
1930	755	2,017,950	1	-	1	2	6,026	.99	2.99
1929	498	1,321,146	-	1	65	66	7,496	49.96	5.67
				PLANT D					
1931	43	133,730	-	-	2	2	13	14.96	0.097
1930	64	202,240	-	1	6	7	86	34.61	.425
1929	74	236,800	-	2	24	26	219	109.80	.925
1928	79	248,850	-	2	13	15	139	60.28	.559
1927	82	255,840	-	2	19	21	231	82.08	.903

Table 3.- Accident frequency and severity data at refineries A, B, C, and D in 1929, 1930, and 1931

Refinery	1931				1930				1929			
	Em-ployees	Man-hours	Number of acci-dents	Days lost	Em-ployees	Man-hours	Number of acci-dents	Days lost	Em-ployees	Man-hours	Number of acci-dents	Days lost
A	3,600	8,674,915	31	15,435	4,556	12,268,656	73	11,152	4,651	12,546,864	102	17,577
B	502	1,165,293	28	359	644	1,317,527	77	455	578	1,402,861	109	583
C	1,256	3,828,090	1	10	755	2,017,950	2	6,026	498	1,321,146	66	7,496
D	43	133,730	2	13	64	202,240	7	86	74	236,800	26	219
Total	5,401	13,802,028	62	15,817	6,019	15,806,373	159	17,719	5,801	15,507,671	303	25,875
Frequency	4.49				10.06				19.54			
Severity	1.15				1.12				1.67			

During the years 1929, 1930, and 1931 these four plants had a total of 524 accidents; 208 occurred from September to February and 316 from March to August. The fewest accidents occurred in December and September and the largest number in March and July. Accidents appear to happen more frequently during the spring and summer months.

In Table 4 the accident rates are given for the four plants by months during the 3 years for which figures were available.

With 303 lost-time accidents in 1929, 159 in 1930, and 62 in 1931, a reduction of approximately 50 percent was realized between 1929 and 1930 and of more than 60 percent between 1930 and 1931. Table 3 shows that the four refineries reduced their accident-frequency rate from 19.54 in 1929 to 10.06 in 1930 and to 4.49 in 1931. The combined accident-frequency rate of 4.49 for the four refineries in 1931 is much better than the frequency rate of 14.14 for the petroleum industry as a whole in 1931 as given in Table 1. The 1931 accident-severity rate of 1.15 for the four refineries is also much better than the 2.06 severity rate given in Table 1 for the petroleum industry as a whole in 1931.

Table 4.- Accidents at four refineries, A, B, C, and D, by months

Month	1931	1930	1929	Total
January	5	14	26	45
February	4	11	23	38
March	4	18	37	59
April	5	15	35	55
May	7	11	28	46
June	5	15	33	53
July	11	20	26	57
August	5	13	28	46
September	4	11	13	28
October	6	12	17	35
November	4	11	20	35
December	2	8	17	27
Total	62	159	303	524

Table 5 classifies these accidents by causes.

Table 5.- Causes of accidents

Cause	Number
Chemicals and hot substances	77
Slipping, tripping, and stumbling	73
Dropping objects handled by injured	70
Explosions and fires	52
Flying objects	45
Falling from height	39
Hand tools	32
Falling objects	22
Moving objects	19
Machinery	17
Operation of vehicles	16
Dropping objects not handled by injured	14
Running into or striking objects	14
Fumes	13
Lifting, pulling, pushing	10
Stepping on objects	8
Electric shock or burns	3
All causes	524

Parts of Body Injured

Of these 524 lost-time accidents, foot and toe injuries lead with 138, eyes are second with 74, legs third with 49, body and face fourth with 45 each, and ankles fifth with 42 injuries.

For ease in visualizing the picture and attempting a probable solution for the various types of refinery accidents, Table 6 gives the number of injuries to the several parts of the body, in order of frequency.

Unquestionably the use of goggles, hard-toed shoes and possibly leggings would have a vital influence in lessening the occurrence of refinery accidents, and also in decreasing the severity of those which might happen.

Table 6.- Parts of body injured

Part	Number	Part	Number
Feet and toes	138	Arms	22
Eyes	74	Scalp	21
Legs	49	Strains	20
Body	45	Knees	18
Face	45	Hands	15
Ankles	42	Internal	13
Fingers	22	Total	524

Summary and Conclusion

The refineries in Louisiana have reduced their accident frequency and severity during the period covered by this report through the combined efforts of operating companies and individual employees.

In 1931, according to the Safety Division, American Petroleum Institute, reports from 89 oil companies employing 227,360 men in all parts of the United States gave the frequency as 14.14 and the severity as 2.06. During the same period in the four Louisiana plants the frequency rate was 4.49 and the severity rate 1.15.

The figures in Table 6 unquestionably indicate that if goggles were worn where the nature of the work requires them, if hard-toed (safety) shoes and leggings were worn for certain classes of work, and if more attention were given proper construction, drainage, storage, and handling materials that now cause slipping, tripping, and stumbling hazards, the accident experience of the Louisiana oil refineries would be very likely to realize a further reduction or improvement in their already good accident experience.

I. C. 6722,
June, 1933.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

TIMBER WITHDRAWING AND DEVICES USED FOR THIS PURPOSE IN SOME COAL MINES[1]

By J. W. Paul[2] and J. G. Calverley[3]

MEANING AND PURPOSE OF TIMBER WITHDRAWAL

Withdrawing timber in coal mines is done for the purpose of inducing the roof to cave in the mined-out areas and thereby relieve stresses in the immediate roof material over the working places adjacent to the breakline of the roof, thus preventing the crushing of the ends of pillars and adding to the safety of those engaged in pillar extraction. A secondary purpose is the recovery of timber for further use either as props, crossbars, lagging, sprags or cap pieces. At some mines timber is withdrawn as a precaution against fire which may be of spontaneous origin, because wood always aids a fire of any origin.

With some of the power-driven types of machines now in use a greater proportion of mine timber may be recovered, and in mines where pillars are left too narrow for their removal some of the timber in rooms might be salvaged for further use.

In all timber withdrawal operations there is the danger of the roof's falling when the timber is withdrawn, thus introducing a hazard to the persons employed in the work unless they exercise all precautions to give themselves protection.

Improper methods of timber withdrawal have resulted in injuries and fatalities to those engaged in such work. This paper brings to attention a number of these methods and gives information as to better devices and practices which are in use at some mines and may be used by others with safety to those engaged in this seemingly hazardous work.

The information given in this paper has been abstracted and compiled from data in Bureau of Mines reports which have been made on the study of roof in mines operating in the Pittsburgh coal bed in western Pennsylvania, Ohio, and West Virginia. Permission was given by the respective operators for the publication of certain features of the confidential reports relating to a study of falls of roof and coal, and this circular is confined to one phase of the study not specifically treated in other publications.[4]

For the past few years much attention has been devoted to the withdrawal and the recovery of mine timber for the purpose of mine safety and economy.

It has been recognized by many managers of bituminous coal mines that the leaving of timbers in the goaf or gob is not only extremely wasteful but is also the cause of roof

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1 — The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6722."

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4 — A study of falls of roof and coal in mines of Harrison County, West Va., Report of Investigations 3110 (1931); A study of falls of roof and coal in mines in the No. 8 Field of Eastern Ohio, Report of Investigations 3070 (1931); Falls of roof and coal in mines operating in the Pittsburgh coal bed in Marion and Monongalia Counties, West Va., Technical Paper 522 (1932); Falls of roof and coal in mines operating in the Pittsburgh coal bed, Panhandle district, West Virginia, Technical Paper 534 (1932); Roof support in coal mines in the Irwin, Greensburg, and Latrobe Basins, Westmoreland County, Pa., Report of Investigations 3113 (1931).

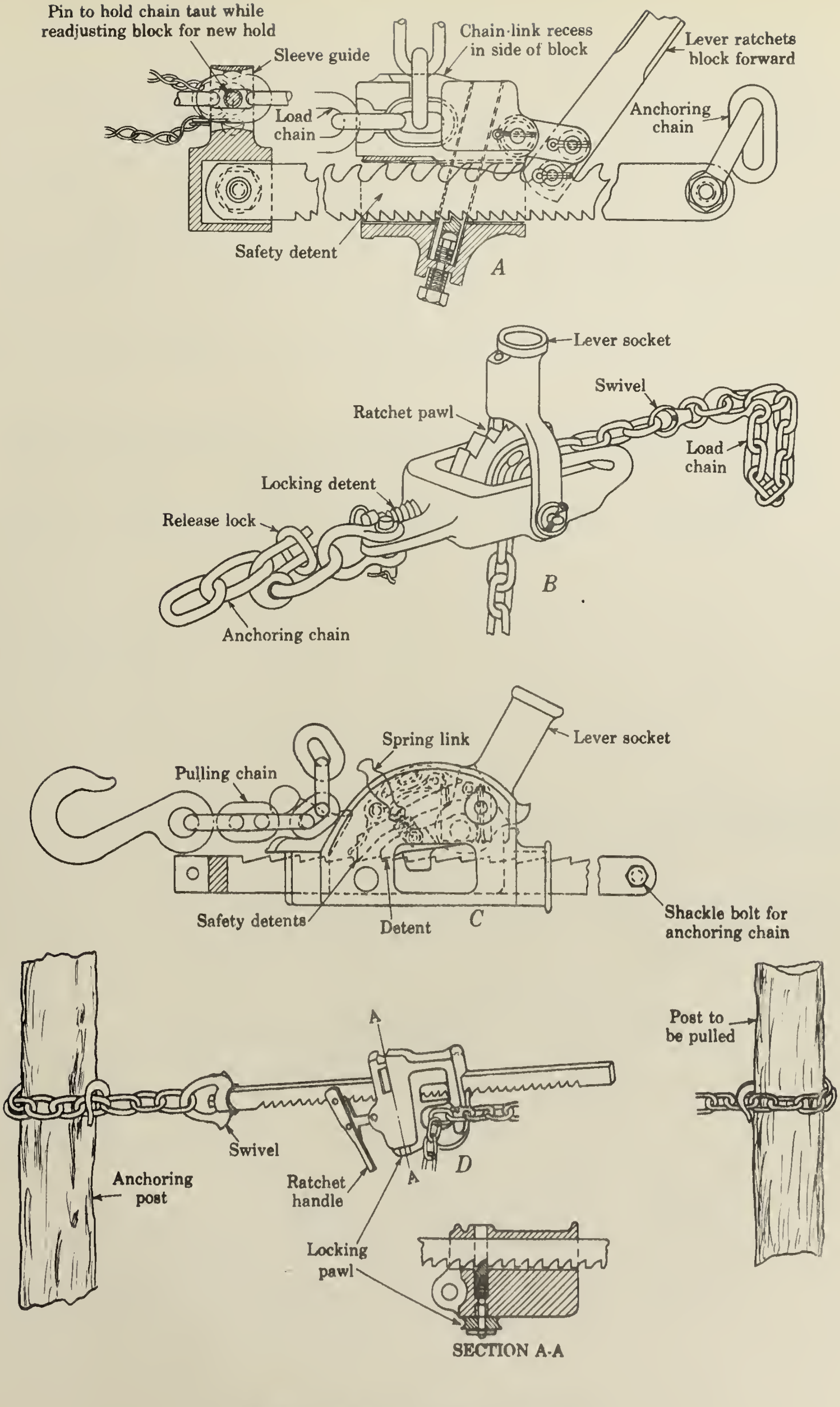


Figure 1.—Types of hand-operated post pullers.

stresses in the mined-out area which affect the roof and coal at the active working face. The practice of leaving the supporting timbers in the goaf or mined-out area is largely responsible for the irregular subsidence of the roof found in some mines which results in irregular fractures in the roof strata and gives rise to weak roof conditions in the working place, thus increasing the roof hazard and necessitating extra timbering.

Inasmuch as the operation of timber removal is of some importance with respect to proper roof control, it is often attended with a great amount of risk to the workmen withdrawing the timbers; consequently, this has led to the introduction of a number of methods and devices to eliminate, as far as possible, the hazards in connection with timber withdrawal. (Fig. 1.)

Following is a brief summary of the timber-withdrawal practice followed at a number of bituminous mines studied:

Timber-drawing methods in certain mines in the Pittsburgh coal bed

Means of withdrawing timbers	Eastern Ohio	Western Pennsylvania	West Virginia Northern	West Virginia Panhandle
Blasting	0	X	X	0
Cutting with ax	X	X	-	0
Chain and bar	0	X	X	0
Post-pulling jack	0	X	-	0
Power-driven post puller	0	X	X[1]	0
Locomotive and cable	0	X	-	0

[1]Recently installed in some mines of the district.

From this tabulation it is shown that six of the methods are practiced in western Pennsylvania, three in northern West Virginia, and one in eastern Ohio. In the Panhandle of West Virginia and in eastern Ohio pillars are not recovered; consequently, the need for timber withdrawal is not a part of the routine of mining.

This investigation covers 41 representative bituminous coal mines, 3 of which are in eastern Ohio, 26 in West Virginia, and 12 in western Pennsylvania. Table 1 gives comparative data relative to the mines and districts considered.

Table 1.- Extent of timber withdrawing and persons killed and injured in selected mines

	Eastern Ohio	West Virginia, Marion and Monongalia Counties	West Virginia Panhandle	Western Pennsylvania
Extent timber is withdrawn to create falls	Slight	Slight	Slight	Extensively
Number of timber-withdrawal fatalities	0	3	0	7
Nonfatal injuries	0	0	0	29
Posts used per 1,000 tons	202	75	221	227
Timber recovered, percent	-	-	-	10-60, posts 65-90, blocks

Comparative Data Relative to Timber Withdrawal

Table 2 shows that 10 timber-withdrawal fatal accidents occurred in two of the four districts considered. In each case the fatalities, from falls of roof, were directly caused by the victim's attempt to withdraw supporting timber in the wrong manner: By bumping the supporting posts loose with another post or weakening the supporting posts with an ax. As shown in Table 2, 29 persons were injured nonfatally while in the act of withdrawing or recovering withdrawn timber by wrong methods; 2 by cutting the post with an ax; 11 by using a post for bumping out the timber; 7 when under loose roof to retrieve timber; and 9 as the result of improper use of post pillars.

Table 2.- Classification of timber-withdrawal accidents, fatal and nonfatal, by causes

District	Weakening post by means of an ax		Bumping post loose with a post		Recovering withdrawn timber		Mechanical post puller		Number of accidents		Is timber withdrawal generally practiced?
	Fatal	Non-fatal	Fatal	Non-fatal	Fatal	Non-fatal	Fatal	Non-fatal	Fatal	Non-fatal	
Eastern Ohio	-	-	-	-	-	-	-	-	-	-	No
West Virginia:											
Harrison County	-	-	-	-	-	-	-	-	-	-	No
Panhandle district	-	-	-	-	-	-	-	-	-	-	No
Marion County	2	0	1	-	-	-	-	-	3	-	To some extent
Western Pennsylvania	2	2	5	11	-	7	-	9	7	29	Extensively
Total	4	2	6	11	-	7	-	9	10	29	-

At the time of this study, timber withdrawal was extensively practiced in western Pennsylvania. The other districts under consideration practiced timber withdrawal to a slight extent. However, the mines in Pennsylvania, Ohio, and the Panhandle district of West Virginia used about the same number of posts per 1,000 tons of coal produced. The mines in Marion County, West Virginia, averaged 75 posts per 1,000 tons.

Table 3 gives the relative rates for all falls of roof accidents at the mines under consideration, based upon thousand 300-day workers, severity, and frequency. The frequency rate is a measure of the occurrence based upon the number injured per million hours of exposure; however, the frequency rate of accidents resulting from withdrawing timber does not bear a similar ratio. In Pennsylvania the frequency rate for accidents in timber withdrawal is 1.27 for the 12 mines listed and for the 6 mines in Marion County, West Virginia, the rate is 0.26. However, no nonfatalities were reported as having occurred in Marion County in timber withdrawal work, which is less extensively practiced there than in Pennsylvania.

Table 3.- Number injured per thousand 300-day workers and severity and frequency rates of falls of roof accidents in the mines studied

Mining district	Number of mines studied	Number injured per thousand 300-day workers	Severity rate	Frequency rate
Eastern Ohio	6	113.08	12.10	43.4
Western Pennsylvania[1]	12	52.95	7.32	22.06
West Virginia:				
Harrison County	5	22.16	4.15	9.85
Monongalia County[2]	7	21.49	5.09	9.69
Marion and Monongalia Counties	9	37.62	6.16	16.55
Panhandle district	6	47.25	9.77	21.05

[1]Timber withdrawal is extensively practiced.
[2]Operated in the Sewickley coal bed.

EASTERN OHIO

Timber Withdrawal

Of six representative mines included in the investigation of timber withdrawal in the eastern Ohio district, in one mine only - a mechanical loader equipped mine - was timber withdrawing practiced; this included not more than the posts in the two rows nearest the roadway.

For a period of 8 months for which a record was obtained, about 297,346 tons of coal were produced and about 64,244 mine props were used. Of this number about 32,122 props, or 50 percent, were recovered by the timber crews with the aid of an ax or a post used as a ram; either of these methods is hazardous and should be abandoned in favor of a mechanical post puller.

The mines included in this group are each developed on the room-and-pillar system. The barrier and room pillars are for the most part left standing because no pillars are withdrawn. These pillars represent a very appreciable percentage of the available coal of the mine and to a large extent are a total loss.

PANHANDLE DISTRICT OF WEST VIRGINIA

At none of the six representative mines in the Panhandle District of West Virginia was the withdrawing of timber from the mined-out area practiced, largely because coal pillars are not extracted.

The mines included in this group are all developed on the room-and-pillar system. The rooms are generally driven 24 feet wide and on 32-foot centers, thus leaving for the greater part an 8-foot thick pillar between rooms. Room stumps and entry chain pillars are not extracted. This method of mining leaves a large percentage of the coal standing.

At a number of the mines of this district the management has made an occasional attempt to extract the pillars and to withdraw the timbers, but the attempts were abandoned because the heavy overlying limestone showed no immediate evidence of fracture. At no mine of the group was there evidence that a systematic attempt had been made to extract pillars in conformity with successful methods used in other districts having similar overburden.

It would appear that had the original room pillars been of sufficient thickness to afford support to the main roof while the adjoining area was being mined out, systematic roof falls could be created, resulting in part pillar recovery.

FAIRMONT DISTRICT OF WEST VIRGINIA

Harrison County

This study includes five mines in Harrison County, W. Va., northern district. Timber withdrawal of any consequence was not practiced at any of these mines.

Pillars were for the most part extracted, but the falls along the fracture line were not created by timber withdrawal.

At two of the mines of this group, company rules stated that, where necessary, timbers should be withdrawn by means of an approved mechanical post puller (fig. 1) operated by men familiar with timber recovery work, who do the withdrawing in the presence of the mine foreman or section boss.

Monongalia County

The data in Table 3 for Monongalia County applies to mines operating in the Sewickley coal bed where timber withdrawal is not practiced.

Marion and Monongalia Counties

At nine mines in Marion and Monongalia Counties, operating in the Pittsburgh coal bed, timber withdrawal was practiced to some extent primarily to induce the roof to break along the fracture line where pillars are withdrawn. The six mines, A to F, inclusive, in Marion County either practice timber withdrawal or have regulations for its withdrawal.

Mine A.- At mine A no timber withdrawal or timber recovery of any consequence was practiced at the time of this study; however, at places where it appeared necessary to remove timbers, the following company rule was rigidly enforced:

> Roof timbers shall be pulled only by mechanical means by men experienced in such work and under the direct supervision of the mine foreman or section foreman. Timbers shall be pulled only at regular intervals and line of retreat shall be maintained at all times.

No timber-withdrawal accidents were reported as having occurred.

Mine B.- At mine B the posts in the mined-out area of the pillar places are recovered by the timbermen with a chain-and-bar type of post puller. A large number of the withdrawn posts are used again for roof support and the broken ones are used for lagging. The number of posts recovered was estimated to be 50 percent.

The timber withdrawal and recovery was accomplished with no reported accidents.

Mine C.- At mine C a few posts are withdrawn by means of a mechanical post puller when necessary to create a fall; however, it was observed that posts in a few of the mined-out places had been cut half through with an ax so that they might be easily broken by the weight of the roof.

The practice of cutting a post to weaken it at this mine was responsible for a fatality. A mechanical post puller was available in the place at the time this accident occurred.

Mine D.- At the time of the study at mine D timber withdrawal generally was not practiced; however, a few props were withdrawn from time to time to bring about a roof fall. Created falls were made, generally, in accordance with the following company rule:

In making falls, the recovery of posts and timbers shall be done only under the supervision of an official or some other competent person and with a post puller or other mechanical device.

Mine E.- Timber withdrawal at mine E was not generally practiced; however, some timbers were withdrawn in the worked-out areas along the fracture line by means of bumping the post to be withdrawn with another post.

During the period of 12 months covered by this study a workman was fatally injured while attempting to bump out a post with another post.

Mine F.- At mine F posts are withdrawn along the break lines as the coal is extracted and for the most part by means of a chain-and-bar device; however, in one instance a fatality occurred because the victim, a timberman, cut with an ax a supporting post to weaken it so it would not retard the caving of the roof.

A company rule states that all timbers be extracted under the direct supervision of a boss or competent timberman and that none be withdrawn by the loaders or inexperienced men.

Figures showing the amount of timber extracted and recovered were not available; however, a high recovery was estimated.

Table 4 gives a summary of timber withdrawal practices of mines A to F, inclusive.

Table 4.- Summary of timber withdrawal practices in mines A to F

Mine	Timber withdrawal	Percent recovery	Method of withdrawal: Definite	Method of withdrawal: Indefinite	Withdrawal rules	Means of withdrawal	Timber withdrawal under supervision	Remarks
A	A few	-	Yes	-	Yes	Mechanical	Yes	Posts drawn when necessary.
B	Yes	50	Yes	-	No	Chain and bar	-	Regular timber crews employed.
C[1]	A few	-	-	Yes	No	Bumping post puller and ax	Yes	Rib boss present when making fall; some timbers weakened by ax.
D	A few	-	Yes	-	Yes	Post puller	No	Post drawn when necessary to create a fall.
E[1]	A few	-	-	Yes	Yes	Bumping with post	No	Posts drawn when necessary to create fall.
F[1]	Yes	Large	Yes	-	Yes	Chain and bar cutting with ax	Yes	Timber drawing is practiced to create a fall.

[1]Fatality occurred chargeable to timber withdrawal.

WESTERN PENNSYLVANIA DISTRICT

In the western Pennsylvania district group of mines - A to L - timber withdrawal is for the most part practiced extensively. At each of the mines controlled, fracture lines are maintained. They vary in length 500 feet to more than 3,000 feet; consequently, timber withdrawal is practiced primarily to assist, as far as possible, in maintaining the break

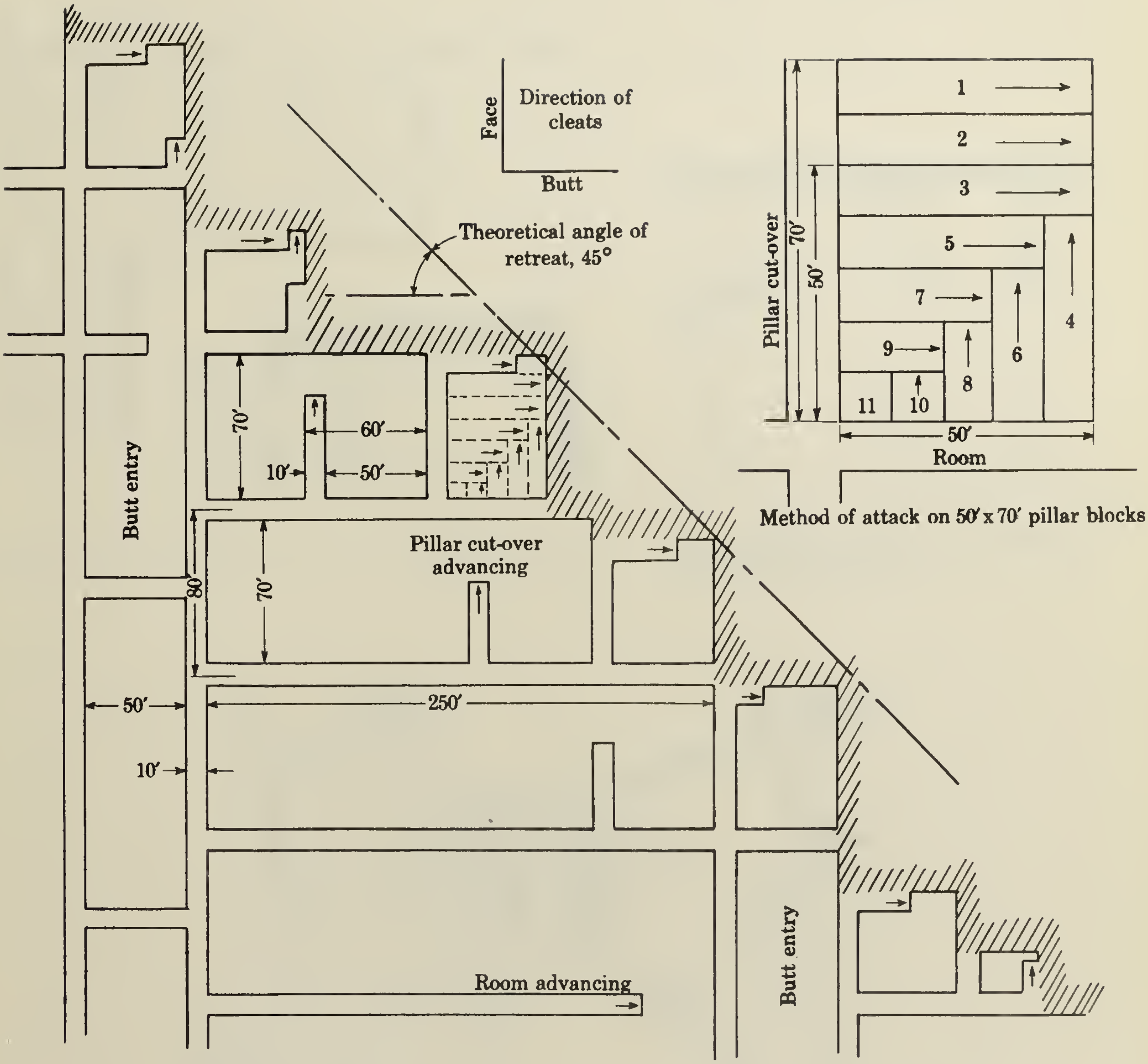


Figure 2.—General method of open-end pillar mining, western Pennsylvania district.

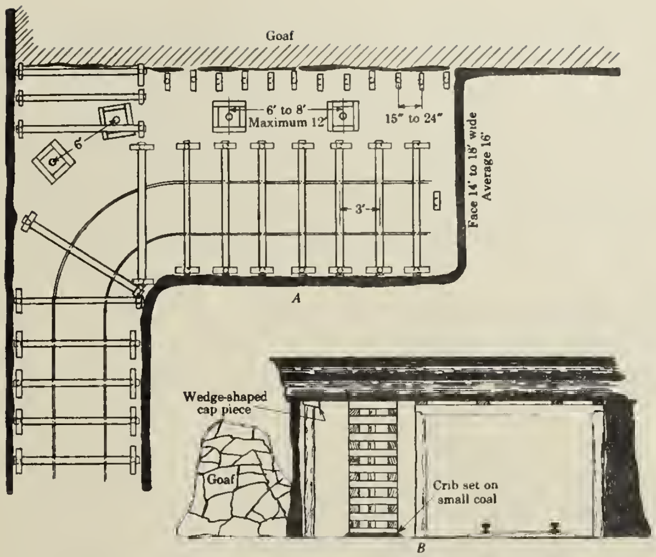


Figure 3.—Details of timbering in pillar work. *A*, Plan of timbering; *B*, section across open end.

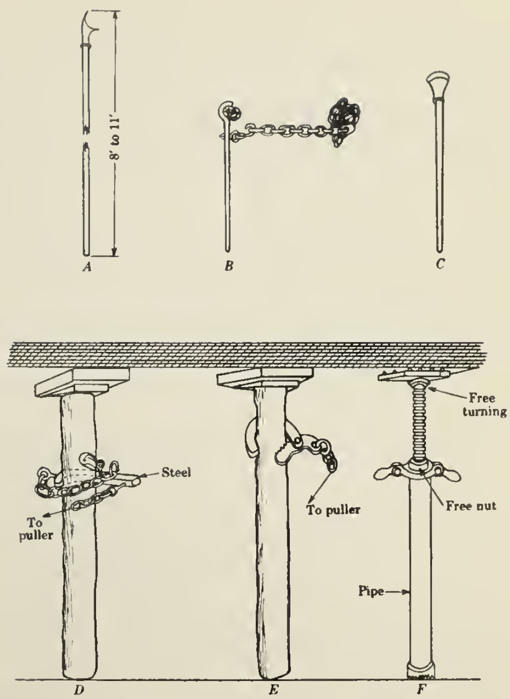


Figure 4.—Timber withdrawal devices. *A*, Timber recovery pike; *B*, bar-and-chain post puller; *C*, ram; *D*, post decapper (tangential); *E*, post decapper; *F*, adjustable post.

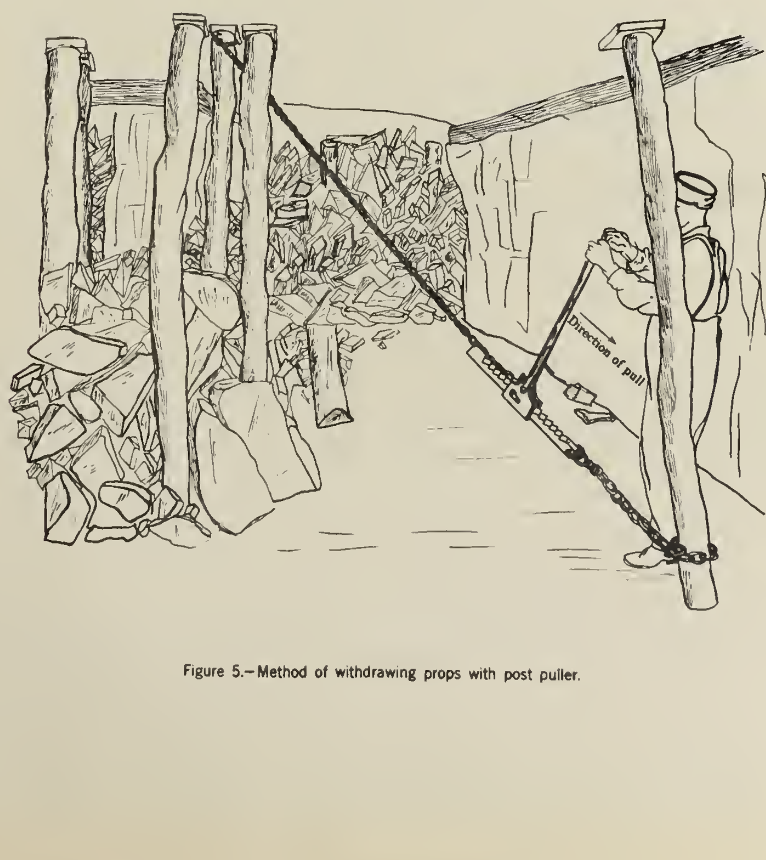


Figure 5.—Method of withdrawing props with post puller.

lines in proper alinement and to prevent the overlying burden from riding over the adjacent active workings with consequent weakening and fracturing of the roof. Figure 2 is a plan of a general method of pillar mining in one district of western Pennsylvania where the break line of the roof is maintained straight by systematic recovery of the pillars by a succession of cuts across the end and side of the pillars, a definite method of timbering, and the systematic withdrawal of the timber following each cutover.

As shown in Table 5, there were 7 fatal and 29 nonfatal injuries from falls of roof while withdrawing timber at the 12 mines, A to L. As previously stated, 5 fatal and 11 nonfatal accidents resulted from bumping a post with another post, 2 fatal and 2 nonfatal accidents from cutting the post with an ax, 6 nonfatal accidents occurred while retrieving posts from under unsupported roof, and 1 nonfatal injury was received while removing a jack.

Table 5.- Cause of timber withdrawal accidents, fatal and nonfatal, mines A to L

Cause	Accidents: Fatal	Accidents: Nonfatal	Accidents: Total
Knocking post loose by means of another post	5	10	15
Recovering withdrawn posts	-	3	3
Using a mechanical post puller improperly	-	9	9
Recovering cribbing blocks	-	3	3
Cutting supporting post with an ax	2	2	4
Bumping crossbar legs loose	-	1	1
Removing an adjustable jack	-	1	1
Total	7	29	36

Following is a brief discussion relative to timber withdrawal of each mine, A to L, of the methods and practices:

Mine A

At mine A the open-end method of mining is the practice. Closely set posts in each working place are carried next the goaf, and in addition to the posts substantial cribs are erected at regular intervals (fig. 3). When the miners have driven the place to completion, they immediately withdraw the timbers with a mechanical post puller in the presence of an underground official, either the assistant foreman or a rib boss. Following are some of the important company timber-withdrawal rules which are strictly enforced by the underground officials:

Rule (a). In the making of falls in pillar workings or drawing of timbers in other places, safety posts shall be set to protect the workmen. Breaker rows shall be set in the roadway at the outer edge of all pillar falls before drawing posts or timbers to make a fall.

Rule (b). The knocking out or cutting of posts with an ax or other hand tools is positively prohibited. Where timbers are to be removed, it shall be done with mechanical post puller or other mechanical equipment only.

Rule (c). Would recommend a long hook for use in recovering posts in making falls. The purpose of this is to withdraw posts already released and in positions too dangerous for workmen to enter to recover them. (See fig. 4 for type of pike pole.)

At the time of erecting the crib blocks, some consideration is given to their removal, and a post is placed in the center of the crib at the time of erection. This post is to afford protection to the workmen while erecting the crib and when the crib blocks are being withdrawn; also, the first or bottom blocks are placed on several inches of small coal to facilitate their removal.

During the period 1928-1931, covered by this study, no fatal accidents caused by timber withdrawal were reported. However, there occurred in the same period 6 nonfatal accidents (with total of 97 days lost time), of which 5 were caused by pulling the lever of the post puller toward the goaf and 1 by a fall of roof while the victim was engaged in attaching the chain to a post to be withdrawn. For the most part, since the occurrence of these accidents, those chargeable to the post puller have been eliminated largely by requiring the operator to pull on the post-puller handle away from the goaf rather than push toward the goaf (fig. 5).

Specific figures relative to the number of posts and blocks recovered when withdrawn were not available; however, it was estimated at 25 percent for props and 85 percent for cribbing blocks.

Mine B

The general plan of mine B is substantially the same as mine A except that the pillars are extracted by the pocket-and-stump method rather than the open-end method as in the case of Mine A. Cribs are not used for roof support; only posts and crossbars are used for this purpose.

When the pockets are driven through the pillar and the stump is removed, the timberman immediately withdraws the supporting timber in dangerous places, using a mechanical post puller in the presence of a fall boss and working under the direct supervision of the assistant mine foreman. Except for the places considered dangerous, the general practice of removing timbers is to loosen the cap piece of the supporting post by means of an ax and then to bump the post loose with another post -- a hazardous practice conducive to injury to the workmen.

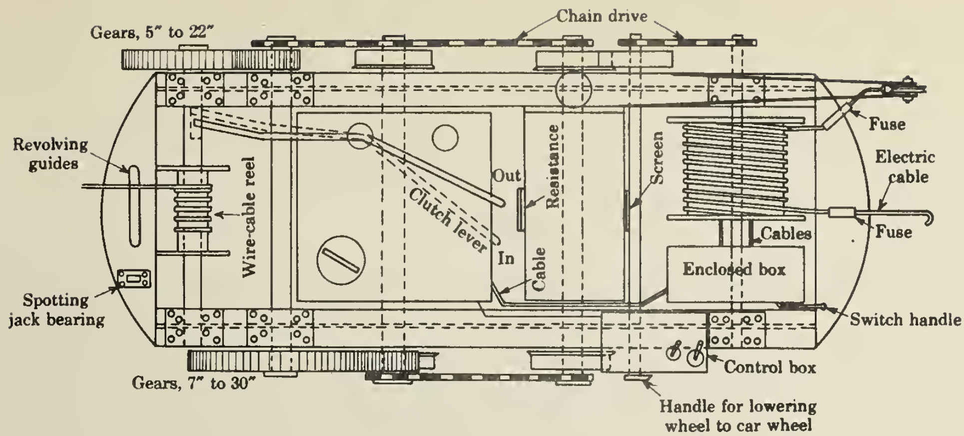
Men have been specially trained for timber withdrawal by the mine foreman, and none other than experienced men are permitted to withdraw timber.

In some extracted areas the withdrawal of posts is not necessary to obtain a rock fall. For the period covered, 1925 to 1928, inclusive, no accidents were reported, fatal or nonfatal, as the result of timber withdrawal.

Mine C

At mine C the timber is generally withdrawn. This induces falls along the pocket-and-stump mined-out area. The timber is extracted largely by means of a mechanical post puller, but some posts are loosened by being bumped with another post--a practice forbidden by the company.

During the period 1929-1930, there occurred 4 timber withdrawal accidents; of these, 3 were caused by withdrawing timber by means other than the post puller, and 1 was caused by a roof fall while recovering a withdrawn post. The accidents resulted in a total of 53 days lost time to the workmen.



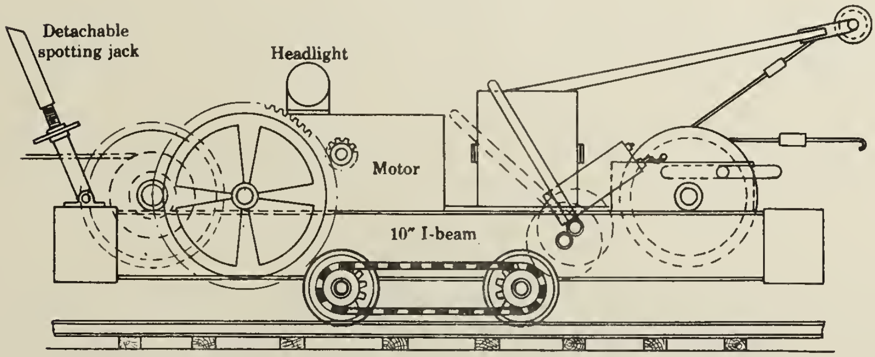


Figure 6.—Electrically driven post puller.

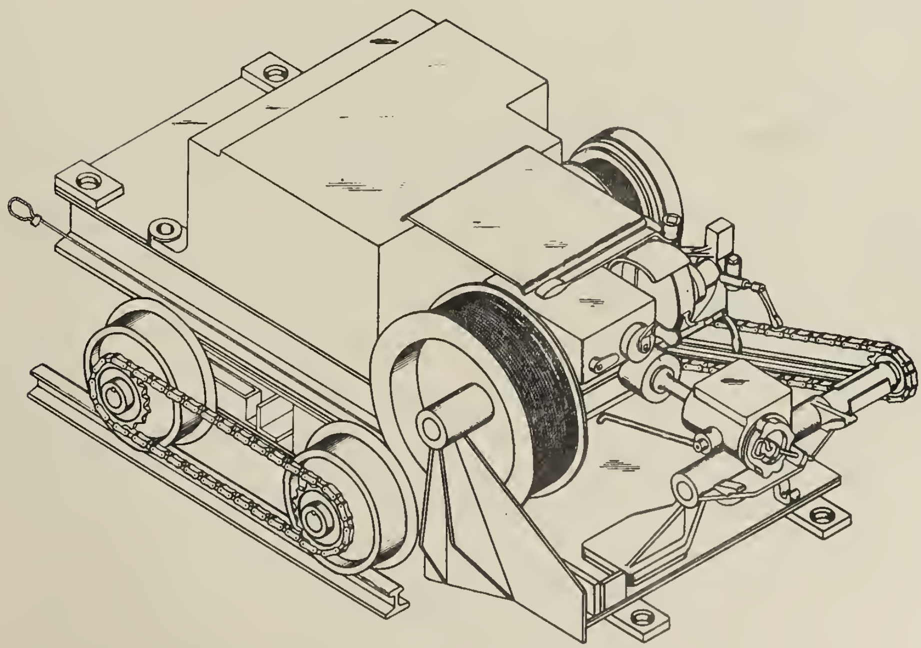


Figure 7.—Mining machine mechanism used as a post puller.

Of the posts used, about 10 percent are regularly recovered and used again.

Mine D

At mine D the withdrawing of timber is extensively practiced along the various pocket-and-stump fracture lines.

At the time of this study the withdrawing was accomplished by specially trained timbermen and generally by the aid of a hand-power mechanical post puller; since this investigation, however, an electrically driven post puller (fig. 6) has to a large extent superseded the bar-and-chain type which was formerly used.

It was learned that the withdrawing of some posts by means other than a mechanical post puller resulted in 5 timber withdrawal accidents, one of which was a fatality caused by the fall which the victim was creating when removing the timber by means of another post used as a ram.

Accurate data relative to the actual number of pieces of timber recovered were not available; however, the percentage was high.

Mine E

At mine E specially trained timbermen use a mechanical post puller to withdraw timbers along the controlled fracture line of the pocket-and-stump mined-out area. The practice of bumping posts loose or hacking them with an ax is strictly prohibited. Over a period of 21 months no injury was reported as the result of withdrawing timber.

Mine F

At mine F timber withdrawing is intensively practiced along all the fracture lines. The withdrawing, generally, is accomplished with an electrically driven, post-pulling machine operated by specially trained timbermen.

At this mine one of the earliest power-driven post pullers was developed from suitable parts of a mining machine (fig. 7); it is novel in that more than one prop can be withdrawn at a time by the use of chains of different length attached to the main cable. This machine has given satisfactory results as to safety and economy.

Prior to the installation of the power-driven machine, the withdrawal was accomplished by a bar-and-chain post puller.

During the period 1928-1930 5 timber-withdrawal accidents occurred, 1 fatal and 4 non-fatal. Four of these were chargeable to the bar-and-chain post puller which was defective at the time of the accidents and the other was due to the victim's withdrawing a post by hand and bringing about a fall of roof.

The percentage of timber withdrawal and recovery at this mine is high since installation of the power-driven post puller. It is recorded that for a 38-month period, 119,324 pieces of material were salvaged without a timber-withdrawal accident. It was also recorded that the operating index of tons of coal produced per prop used increased from approximately 4 to 7.

Mine G

At the time of the study at mine G, generally no attempt was made to withdraw posts from the extracted area along the pocket-and-stump fracture line; since that time, however, an electrically driven post puller has been installed and the practice of post withdrawal has given creditable results as to safety and economy (fig. 7).

During 1929-1930, one timber withdrawal accident occurred when bumping out a post caused a roof fall, resulting in 18 days' lost time to the victim.

Inasmuch as the general mining conditions and practices are about the same as those at mine F, the ratio of timber withdrawal and recovery will be approximately the same since the installation of the power-driven post puller.

Mine H

At mine H the open-end method of mining is used which is similar to the practice which prevails at mine A of this group; consequently, systematic timber withdrawal is followed in order that a controlled fracture line may be maintained along the goaf of the mined-out area and the active working face.

The timbering consists of posts and substantial block cribs (blocks 5 by 5 by 30 inches). It was observed that no consideration was given at the time of erecting the cribs to providing for their facile and safe withdrawal as were the practices at mine A. No posts were used in the center, adjacent to, or near the crib to afford roof support while the crib was being dismantled at the time of withdrawal, nor were the cribs, generally, built on a few inches of small coal; these practices are not conducive to safe timber withdrawal (fig. 8).

When falls are being created along the fracture line, special timbermen withdraw and recover the timber. The timber generally is withdrawn by a mechanical post puller of the bar-and-chain type. It was observed, however, that in some cases the tight posts were hacked with an ax or bumped loose with another post; either of these practices is hazardous.

During a period of 44 months there were 5 nonfatal timber withdrawal accidents; 2 of these were caused by the victim's bumping the supporting posts loose and the others were caused by falls of the roof material while the victims were engaged in recovering the dislodged timbers. These accidents resulted in 637 lost days to the workmen.

About 25 percent of the posts and about 60 percent of the crib blocks are recovered by the timbermen.

Mine I

At mine I timber withdrawing is not practiced generally; however, some posts are withdrawn by the miners by means other than a mechanical post puller and for purposes other than creating roof falls.

During the 12-months period covered by this study, 2 fatal accidents occurred; one of these was caused by the victim's cutting a tight post loose by means of an ax, and the other by the victim's bumping a post out with another post, the roof falling in each case.

Mine J

At mine J the open-end method of mining the pillars is practiced. Closely set posts in each working place are carried next the goaf, and in addition substantial block cribs are erected at 8 to 10 foot intervals. The bottom blocks of the crib are placed on small coal and the crib is erected around a substantial post (fig. 8, A).

As soon as the place is driven to completion, timber withdrawal is commenced by the miners, who generally use a mechanical post puller and work under the direct supervision of a fall boss or special timberman.

During the period 1928-1930, 2 timber-withdrawal fatalities occurred which were caused by the victim's bumping out the supporting legs of a collar, which permitted the roof to fall upon him.

Of the props and cribbing blocks used, 60 percent and 90 percent respectively are recovered.

Mine K

At mine K the mining and timber-withdrawal practices are substantially the same as those of mine J.

In the period, 1928 through the first 4 months of 1932, inclusive, there were 7 timber-withdrawal accidents; one of these was fatal and was caused by a fall of roof while the victim was making an effort to weaken a supporting post by means of an ax to facilitate its removal.

Two of the nonfatal accidents were caused by falling roof while the victims were bumping a supporting post loose with another post, resulting in a total of 137 days lost time. Two of the accidents were caused by the workmen removing the supporting post and not erecting temporary posts before dismantling cribs and collar sets. One accident occurred while the victim was engaged at placing the post puller chain around a post to be withdrawn; a roof fall injured him. The other accident occurred while the workman was withdrawing an adjustable post jack (fig. 4, F). The accidents resulted in a total of 1,472 days lost time.

Of the posts and crib blocks used, about 60 percent and 85 percent respectively were recovered.

Mine L

At mine L the withdrawal of timber along the fracture lines of the mined-out areas is followed intensively.

Fracture lines as long as 1,500 feet are maintained. Immediately on completion of respective pocket-and-stump pillar places the systematic withdrawing of timbers is commenced. Timber withdrawal is accomplished with a mechanical post puller operated by miners who are directly in charge of a specially trained fall boss and in the presence of an assistant foreman or section boss.

In the 12-month period covered by the study at this mine no timber withdrawal accidents occurred.

Specific data relative to amount of timbers recovered were not available; however, the percentage of recovery was reported to be high.

WITHDRAWAL METHODS AND DEVICES USED

Butting Out With a Prop

Table 2 classifies 39 timber-withdrawal accidents, of which 10 were fatalities; of these fatalities, 6 were charged to falls of roof caused by the victim's bumping the supporting posts loose with another post to create a roof fall. Eleven persons were nonfatally injured while similarly engaged.

The practice of bumping posts out by means of another post is especially hazardous, inasmuch as the person bumping loose the supporting post is of necessity directly under the area of the immediate roof which is being supported by the timber about to be withdrawn.

Cutting With an Ax

In some mines included in this study it was observed that in order to assist the roof of the mined-out area to fall, the roof-supporting posts were weakened by cutting a portion

of the post with an ax. This is a hazardous practice and should be prohibited. Table 2 shows 4 fatal and 2 nonfatal accidents as the result of this practice.

In addition to the immediate danger of the falling of the roof which is supported by the timbers that are being removed, the effective forces of the falling roof are probably transmitted through the roof area for several feet from the part from which the timbers have been removed. Men withdrawing timbers should therefore always protect themselves by setting posts where necessary.

It is of interest here to mention a fatality, caused by falling roof material, that occurred in one of the mines in the group. The fall was attributed to the fact that the material was dislodged by forces which were transmitted through the roof by a created fall 20 yards distant from where the fatality occurred. The victim had purposely retreated some distance from his working place to avoid the danger of being struck by material of the created fall.

Removing Material From the Lower End of the Post by Pick

At some mines when supporting posts are about to be withdrawn it is the practice for the miners engaged at this work to loosen the material from around the foot of the post with a pick rather than to decap the post or loosen it at the top. The first method is preferable to that of picking or cutting loose the supporting post's cap, because the immediate roof is not likely to be subject to as much disturbance with consequent hazards of falling loose material from the immediate roof directly above the supporting post. Removing material from the bottom of a post should be preliminary only to its removal by some mechanical means.

Knocking Out Post With a Sledge or Ax

The practice of hammering supporting posts loose by means of an implement such as an ax or sledge is followed at a number of mines. This is a precarious practice and should be prohibited because it has been the direct cause of a number of falls of roof accidents which have resulted in fatalities.

Blasting Out Post

At some mines, supporting posts about to be withdrawn are blasted out of a mined-out area when falls are being made. The blasting out of the posts is especially hazardous because of the possibility that the blast will affect the permanent timbers and the immediate roof of the adjoining active workings so as to cause roof material to fall. Its use introduces avoidable potential hazards, including fire, which might result in an explosion of accumulated coal-dust or gas.

Mechanical Devices

Cable and Ratchet Device.- Timber withdrawal in the mines included in this study was generally by means of a mechanical post puller, bar, rack, or chain-wheel ratchet with chain (fig. 1, A, B, C, and D).

It is worthy of mention that of the 39 classified timber withdrawal accidents, 10 of which proved fatal, no fatalities were charged to mechanical post puller methods; and, of the 9 accidents charged to the mechanical post puller method, all were of a minor nature and could have been avoided with the proper precaution. Some of the causes leading to these injuries were using apparatus with defective mechanism; not fastening securely the mechanical

pulling device to the anchoring post; and the puller operator's standing too near and in front of the puller handle and the anchor post rather than to the rear of the puller handle and to the side of the anchor post (fig. 5). Of the 9 accidents referred to, all were due, largely, to the victims' disregard for the essential precautions.

Locomotive and Cable.- At some mines use is made of a haulage locomotive to withdraw timbers from the mined-out areas. Of the various types used, the hoist-equipped locomotive is the one most adaptable for timber-withdrawal because it may be maintained in a fixed position and the hoist used to advantage to pull out the timbers. In this case the operator is better able to see the withdrawal operation, but when the locomotive and cable only are used, the locomotive is of necessity moving away and consequently the operator is out of view of the withdrawal.

Power-driven Post Puller.- Timber extraction at a few mines where it is extensively practiced is accomplished by means of a self-propelled machine, designed especially for the purpose of timber withdrawal (figs. 6 and 7).

These machines consist, for the most part, of the discarded parts of various types of mining machines, such as truck, motor, and drums; however, in some cases, additional parts are added, such as adaptable gear wheels, clutch, drum, and drum brake.

Essentially these post-pulling machines consist of a series-wound, d.c. motor equipped with the necessary hoist parts mounted on a self-propelled truck with anchoring jacks (hinged fixed), usually four in number, one at each corner of the truck. Figure 6 illustrates some of the essential parts used on a certain mining machine which has been reconditioned for the purpose of withdrawing timbers.

The power-driven post puller has proved highly satisfactory as to safety and economy. At one of the mines included in this study and at which a power-driven machine was being used, several hundred thousand posts have been withdrawn, and the recovery has resulted in a timber-cost reduction of about 2 cents per ton; in accomplishing this recovery, no accident occurred. Timber withdrawal accidents are largely eliminated when the power-driven post puller is used, particularly those accidents due to falling roof material resulting from withdrawal of the supporting posts.

With a power-driven machine's pulling cable to which are attached several post hitching-chains of different lengths, it is possible to withdraw several posts in one operation. The immediate roof from which the timbers are extracted is weakened as each supporting post is withdrawn, thus resulting in the created fall. This feature has the advantage that the persons engaged at withdrawal need not be near the withdrawal area each time a post is about to be removed, as is the case when the posts are drawn one at a time.

CONCLUSIONS

1. Any method of timber withdrawal employed should permit the workmen to be in a safe place while the timber is being withdrawn.

2. Mechanical power-driven devices may justify the withdrawal of timber from mines operating in thin coal which is too expensively removed by hand-power devices.

3. Mechanical, power-driven, timber-withdrawing machines may be assembled largely from discarded parts of coal-cutting machines.

4. It should be noted that where it is the practice to withdraw timber, most of the companies have definite rules forbidding cutting with an ax or knocking out by a ram or hand tool, and that in each case of a fatality this rule was disregarded. This fact brings to attention the necessity of enforcing more rigid discipline among those engaged in withdrawing timber.

5. The advantage of the practice of withdrawing timber is twofold: First, it aids in establishing a uniform break in the lower roof members which induces the main roof to break and subside; second, it tends to prevent the roof over the working places from breaking, thereby reducing the hazard from falling material, less of which will have to be handled or loaded for disposal.

6. Withdrawing timber systematically along the pillar-extraction line aids in establishing a break line in the roof, uniform subsidence of the roof, avoids the crushing of pillars, and affords a high degree of extraction, which is in the interest of conservation and economy.

7. A corollary to systematic withdrawal of timber is a systematic method of placing timber for support of the immediate roof as the advance work progresses.

TIMBER-WITHDRAWAL RECOMMENDATIONS

1. The withdrawal of timber in mined-out areas should be attempted only in the presence and under the direct supervision of the mine foreman or one of his assistants who is experienced and thoroughly familiar with timber withdrawing.

2. For the protection of persons engaged in the work, the immediate roof near the place where timber is being withdrawn should be temporarily supported by some suitable means, preferably by adjustable posts, jack type (fig. 4, F).

3. Withdrawing timber should be accomplished by mechanical means only.

4. The uncapping of timbers should be accomplished by mechanical means, tangential pulling device (fig. 4, D).

5. The pulling chain or cable should be of sufficient length not to necessitate the placing of the post pullers too near the edge of the proposed break line of the roof.

6. Post-puller operators should be required to manipulate the machines from a position in the rear of the machine and not from the front - which means not between the post puller and the gob (fig. 5).

7. The use of power-driven post pullers which are capable of withdrawing several posts in one operation and dragging them to a place where they may be salvaged, should be required in the interest of increased safety.

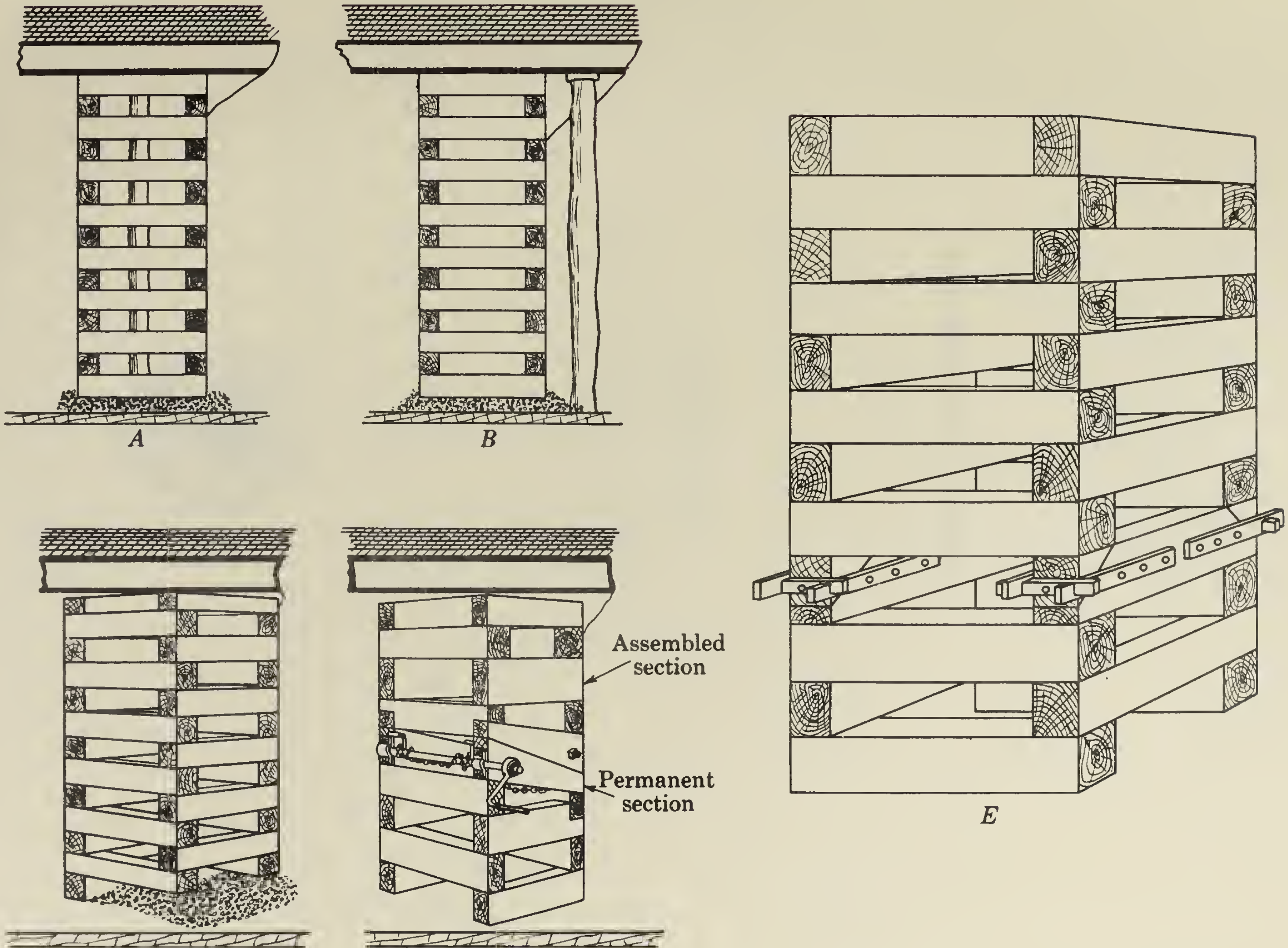


Figure 8.—Types of cribs used in open-end work and in shortwall mining. *A*, Crib with center post; *B*, crib with post adjacent; *C*, crib resting on loose coal; *D*, built-up crib with wedge contact posts for adjustment; *E*, proposed temporary crib.

I. C. 6723 June, 1933

Department of Commerce

United States Bureau of Mines
Scott Turner, Director

INFORMATION CIRCULAR

LIMESTONE

PART I - GENERAL INFORMATION

BY

Oliver Bowles and D. M. Banks

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

LIMESTONE[1]

PART I - GENERAL INFORMATION

By Oliver Bowles[2] and D. M. Banks[3]

CONTENTS

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6723."
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INTRODUCTION

Limestone is the most widely used of all rocks and is essential to a great many industries, including the building trades, highway construction, metallurgy, agriculture, and many chemical and manufacturing enterprises. It occurs in some form in every State and is produced in thousands of quarries. Production in 1930 exceeded 134,000,000 tons. It has more numerous and more diversified uses than any other rock.

This paper is designed to furnish general information on the various branches of limestone quarrying and their economic importance. Brief reference is made to dimension stone, and a more complete discussion follows of the uses, requisite qualities, quarry methods, and processes of preparation of crushed and broken limestone. Footnote references serve to guide the reader to more detailed information.

DESCRIPTION

Limestone is composed ordinarily of calcareous shells of ancient sea animals. It was formed in beds on the sea floor by the accumulation of such shells, together with varying quantities of other substances such as magnesium carbonate, silica, clayey material, salts of iron, and organic matter; and by pressure the entire mass was gradually consolidated into rock. Through long geologic ages of surface adjustment these formations came to occupy land areas and so became available for commercial use.

The essential constitutent of limestone, therefore, is calcium carbonate ($CaCO_3$), and many commercial limestones consist almost entirely of this material. Such rocks are designated commercially as high-calcium limestones. When other substances occur in the stone, classification is made according to the nature of the impurities. For instance, if 10 per cent or more of magnesium carbonate is present, it is called "magnesian" or "dolomitic" limestone; when the amount approaches 45 per cent, the rock is known as dolomite, which is the double carbonate of calcium and magnesium expressed by the formula $CaCO_3 \cdot MgCO_3$. Other varieties are "siliceous" or "cherty" limestones, which contain considerable silica; "argillaceous," containing clay or shale; "ferruginous," of considerable iron content, which usually gives the rock a buff, reddish, or yellowish color; "carbonaceous" or "bituminous," in which carbonaceous matter, such as peat or other organic materials, is present and gives to the rock a dark gray or black color.

Some limestone beds have been formed under conditions that have left many shells almost intact or, at least, in fragments sufficiently well preserved to indicate their origin and character. These are known as "fossiliferous" limestones. Some of them are made up almost entirely of shells of one kind and are named accordingly. "Coral," "crinoid," and "coquina" are common types; "chalk" is a fine-grained, white, friable limestone which is composed largely of minute shells of foraminifera; oyster-shell beds are quite extensive.

Two varieties of limestone, travertine and Mexican onyx, consist of calcium carbonate that has been deposited from solution. The first is a product of chemical precipitation from hot springs. It is deposited in successive layers, and as chemical composition and conditions of deposition may vary during this process a banded structure commonly results. The rock is characterized by numerous irregular cavities. The second type, Mexican or cave onyx, is deposited from cold-water solutions in limestone caves. It is to be distinguished from true onyx, which consists of silica. As Mexican onyx is susceptible of polish it is classed with marbles rather than with limestones.

Another series of names is applied to limestones according to their physical character. "Common compact" is the most widespread type; it consists of a fine-grained, dense, homogeneous aggregate varying in color from light gray to almost black. "Lithographic" is an extremely fine-grained crystalline variety, usually of drab or yellowish color. "Oolitic" limestone is made up of small, rounded grains of lime carbonate with a concentrically laminated structure.

PRODUCTION BY STATES

Deposits of limestone occur in every State. Stone suitable for building is quarried in about half of them, but most of the output originates in Indiana. Quarries for production of crushed or pulverized limestone are very widely distributed; in some States hundreds of them are in operation. While building stone production is a long-established industry, the manufacture of crushed stone has within the past 50 years grown from a small and insignificant industry to one that attained in 1929 a volume of approximately 151,000,000 tons a year. The production (sales) of limestone by States for 3 years is shown in table 1.

Table 1.- Limestone sold or used by producers in the United States, 1929-1931, by States[1]

(Quantities approximate)

State	1929 Short tons	1929 Value	1930 Short tons	1930 Value	1931 Short tons	1931 Value
Alabama	1,190,770	$1,771,201	905,760	$1,537,680	514,290	$1,406,863
Arizona	162,020	147,619	75,390	61,417	31,780	31,502
Arkansas	132,760	170,287	314,250	373,059	87,250	101,438
California ...	261,280	653,681	296,180	587,387	240,180	549,792
Colorado	438,980	458,983	332,310	324,851	233,780	293,660
Connecticut ..	37,030	112,891	35,210	103,359	37,270	95,901
Florida	2,258,130	1,669,245	1,796,670	1,490,173	1,359,460	1,219,214
Georgia	401,320	488,700	315,850	378,060	746,710	658,544
Hawaii	2,670	9,352	2,720	9,108	970	3,454
Idaho	29,820	70,623	19,320	25,099	7,820	14,890
Illinois	8,345,080	6,965,264	7,538,810	5,909,089	5,278,170	3,945,064
Indiana	5,129,220	22,191,883	4,515,460	19,245,409	2,831,910	10,257,555
Iowa	1,624,400	1,549,743	1,813,860	1,850,823	1,271,310	1,208,755
Kansas	1,392,230	1,413,146	1,248,670	1,245,088	1,099,400	1,035,663
Kentucky	2,443,880	2,558,068	2,625,080	2,602,022	2,179,530	1,897,793
Louisiana	(2)	(2)	(2)	(2)	(2)	(2)
Maine	83,980	130,125	101,950	128,892	57,590	82,953
Maryland	346,810	470,246	549,320	723,065	509,180	566,495
Massachusetts.	83,930	303,795	78,730	276,501	69,480	255,080
Michigan	13,572,010	8,425,261	10,432,950	6,596,713	6,059,770	3,805,607
Minnesota	280,690	830,122	300,000	840,860	284,960	816,484
Mississippi ..	(2)	(2)	(2)	(2)	(2)	(2)
Missouri	4,093,430	5,704,241	3,654,920	4,819,475	3,350,660	3,962,469
Montana	134,490	140,164	94,050	114,819	78,740	97,454
Nebraska	178,200	335,789	103,120	188,674	74,030	117,611
Nevada	33,400	51,805	30,120	46,850	(2)	(2)
New Jersey ...	166,750	368,504	176,920	386,451	137,820	329,456
New Mexico ...	(2)	(2)	(2)	(2)	195,550	380,440
New York	11,105,850	13,841,827	11,185,920	13,892,218	10,860,980	12,586,558
North Carolina	101,180	202,416	(2)	(2)	(2)	(2)
Ohio	14,241,690	11,789,747	12,482,760	9,285,089	8,886,640	5,958,261
Oklahoma	2,074,990	1,552,178	1,942,750	1,505,716	1,534,410	1,203,166
Oregon	23,800	58,442	(2)	(2)	(2)	(2)
Pennsylvania..	14,525,080	14,024,287	12,100,700	12,375,067	8,350,060	8,552,692
Puerto Rico ..	55,960	90,675	37,760	85,324	31,450	55,526
Rhode Island..	380	1,171	300	758	60	174
South Dakota..	40,590	55,513	36,770	39,705	39,370	37,320
Tennessee	2,249,440	2,304,002	1,640,530	1,688,763	1,462,600	1,535,836
Texas	3,238,760	2,390,235	2,007,160	1,646,694	1,125,790	951,775
Utah	203,530	224,097	631,420	368,992	164,450	173,362
Vermont	42,360	129,251	39,360	114,053	39,660	115,240
Virginia	2,516,120	2,481,192	2,925,560	2,868,938	2,201,430	2,115,133
Washington ...	75,440	129,423	119,140	165,809	131,600	213,916
West Virginia.	3,301,160	2,978,873	3,008,880	2,653,954	2,017,570	1,768,278
Wisconsin	3,441,070	3,882,645	2,712,640	2,677,898	2,231,190	2,142,927
Wyoming	322,260	473,678	207,380	343,637	166,110	232,892
Undistributed.	304,020	305,681	304,790	426,566	770,060	1,097,393
Total ...	100,686,960	113,906,071	88,741,440	100,002,114	66,751,040	71,875,886

1 Limestone used in cement and lime manufacture not included.

2 Included under "Undistributed."

BRANCHES OF THE INDUSTRY

The limestone industry is divided into two distinct branches - dimension stone and crushed stone. The outstanding distinction is that dimension stone is prepared for use in blocks of definite shapes and often in specified sizes. Rough building stone may consist of quite irregular blocks, but usually at least one good face is demanded. On the other hand, crushed or broken stone consists of irregular fragments.

The two branches have little in common. In quarrying dimension stone, explosives are employed very sparingly because the integrity of blocks must be preserved, but heavy charges of dynamite are used in the crushed-stone quarry to blow the rock into fragments. All subsequent methods and equipment for preparing the products for market are entirely dissimilar. Marketing problems are necessarily diverse.

The crushed and pulverized limestone industries are again subdivided into three main branches, one including stone sold or used in its natural state except for crushing, grinding, and sizing; and the other two involving the manufacture of cement and lime - the important primary products manufactured in conjunction with quarrying operations. The amount of limestone used for all purposes for a series of years is shown in table 2. The first item of this table including all limestone sold or used except that employed in cement and lime manufacture is subdivided by uses for a period of 3 years in table 3.

Detailed statistics of the limestone industry and also of the lime and cement industries are published annually by the Bureau of Mines. These reports may be procured from the Superintendent of Documents, Washington, D.C.

Table 2.- Limestone used for all purposes in the United States, 1927-1931, in short tons
(Quantities approximate)

Use	1927	1928	1929	1930	1931
Limestone (as given in table 3)	99,662,270	96,864,650	100,686,960	88,741,440	66,751,440
Limestone used in cement (including "cement rock") ..	44,195,000	45,012,000	43,612,000	40,841,000	31,736,000
Limestone used for lime manufacture.	8,830,000	8,920,000	8,540,000	6,780,000	5,420,000
Total	152,687,270	150,796,650	152,838,960	136,362,440	103,907,440

Table 3.- Limestone sold or used by producers in the United States, 1929-1931, by uses[1]
(Quantities approximate)

Use	1929		1930		1931	
	Short tons	Value	Short tons	Value	Short tons	Value
Building stone 2/..	1,312,820 (17,864,700 cu. ft.)	$20,649,257	1,152,780 (15,682,720 cu. ft.)	$18,535,293	872,630 (11,706,840 cu. ft.)	$10,858,697
Curbing, flagging, and paving	37,940 (471,880 cu. ft.)	158,266	27,760 (346,040 cu. ft.)	137,801	15,220 (166,260 cu. ft.)	85,175
Rubble	352,480	693,678	756,470	623,100	229,510	296,426
Riprap	2,080,580	2,655,374	2,918,110	3,318,084	2,508,160	2,763,571
Crushed stone	61,465,200	58,934,450	56,775,060	54,154,317	46,601,900	42,752,286
Fluxing stone	24,337,280	17,994,110	17,201,350	12,312,602	9,674,800	7,160,630
Sugar factories ...	487,990	874,909	414,340	733,359	453,640	748,682
Glass factories ...	257,370	455,367	224,180	349,771	159,220	244,466
Paper mills	273,880	456,251	248,790	377,059	194,310	288,143
Agriculture	2,654,580	3,764,775	2,542,100	3,309,329	1,421,050	2,117,141
Alkali works	5,004,930	3,234,457	4,436,160	2,759,245	3,340,290	2,186,422
Asphalt filler ...	344,880	1,165,538	430,290	1,122,939	247,450	778,045
Calcium carbide works	339,510	236,412	364,750	275,296	164,180	96,158
Carbonic acid works	4,560	10,708	2,290	5,700	2,670	5,698
Coal-mine dusting ..	34,650	121,712	47,750	132,985	38,040	117,715
Fertilizer filling ..	3,910	8,656	12,240	21,194	(3)	(3)
Filter beds	72,870	63,277	30,860	23,633	56,100	53,195
Magnesia works (dolomite)	84,750	129,383	111,740	189,219	80,820	122,525
Mineral food	25,270	96,009	30,350	96,216	21,880	93,310
Mineral (rock) wool	83,920	92,092	64,850	70,988	73,640	67,393
Poultry grit	34,600	221,610	45,920	197,327	31,320	159,971
Refractory stone (dolomite).	516,400	461,444	453,350	356,025	268,500	183,020
Road base .	494,250	213,802	139,030	34,106	(3)	(3)
Roofing gravel ...	4,950	10,968	1,740	5,154	(3)	(3)
Stucco, terrazzo, and artificial stone	85,540	292,313	59,570	208,990	45,760	122,181
Whiting substitute	125,430	626,692	119,350	492,894	73,420	408,880
Other uses[4]	166,420	284,561	310,260	159,488	176,530	166,156
	100,686,960	113,906,071	88,741,440	100,002,114	66,751,040	71,875,886

1 Limestone used in cement and lime manufacture not included.
2 Figures for building stone include small amounts of monumental stone.
3 Included under other uses.
4 Includes stone for ammonia, baking powder, dye works, lime burners, nitrates, phosphates, powder, purification of copper, soap, sulphuric acid, and uses not specified.

DIMENSION STONE

While innumerable limestone occurrences are to be found throughout the country, only a few of them consist of rock that will satisfy the exacting requirements of dimension stone. Deposits with irregular or closely spaced joints are not suitable, as only large blocks free from cracks or lines of weakness are in general demand. Limestones display great variation in physical properties such as texture, porosity, hardness, strength, and color; and upon these properties depend their adaptability and value as dimension stones. Only those limestones that are compact, of easy workability, uniform texture, and attractive color are worthy of consideration for this use. Chemical composition is of little significance.

Dimension stone is used principally for building purposes. It may be used for entire exteriors or in conjunction with brick or other building materials. It is also used widely for columns and for interior structural and decorative purposes. The district surrounding Bedford and Bloomington, Ind., produces more than 80 percent of all the building limestone quarried in the United States. It is produced also in Alabama, Kentucky, Minnesota, Texas, Colorado, and Florida and in small amount in a few other States. A discussion of the occurrence, properties, quarrying, milling, and manufacture of dimension limestone will appear in a forthcoming report.

CRUSHED AND PULVERIZED LIMESTONE

Extent of the Industry

Crushed and pulverized limestone found limited use many years ago, but the intensive building activity, public works and highway construction, and industrial development that have characterized recent years have opened up many new and extensive channels of use for this most adaptable of all rocks. The enormous tonnage of crushed and broken limestone consumed in industry is shown in table 4.

Table 4.- Crushed and broken limestone sold or used by producers in the United States, 1921-1931[1]

Year	Short tons
1921	74,358,800
1922	95,184,780
1923	118,342,190
1924	120,506,160
1925	134,220,530
1926	141,321,640
1927	151,163,700
1928	149,025,390
1929	151,135,720
1930	134,425,430
1931	102,789,680

1 Includes stone used for cement and lime manufacture.

Uses

Beginning with the fourth item, riprap, of table 3, the principal uses of crushed, broken, or pulverized limestone are enumerated, together with the tonnage and value for each over a period of 3 years. These figures indicate the relative importance of the various uses as market outlets for limestone. The specific requirements for each purpose are covered in following pages.

For many of the uses of stone in crushed form, physical properties are of primary importance; for others, certain chemical compositions are requisite. Limestone possesses the physical qualities that make it adaptable for practically all the uses for which any crushed stone may be employed and also has active chemical properties that make it not only useful but absolutely essential to a great many industries.

Uses for Which Physical Properties are Most Important

Concrete Aggregate.- A vast tonnage of limestone is used as aggregate for concrete, in which form it is consumed chiefly in highway construction and in the building trades.

While consumers differ as to the requisite qualities of material for aggregate, generally it should consist of clean, hard, strong, durable, uncoated fragments free from injurious amounts of soft, friable, thin, elongated, or laminated pieces. Alkalies and organic matter are usually undesirable. Soluble sulphides are objectionable because they oxidize; the sulphuric acid formed attacks any calcareous material or lime in the cement and forms gypsum, which expands during crystallization and disrupts concrete.

Various tests are made to ascertain the fitness of limestones. They include the Deval abrasion test, the Dory hardness test, the Page impact test, and the ordinary methods of crushing-strength tests. For determining soundness the more important are the freezing and thawing, the sodium-sulphate, the sodium chloride, and the alkali tests; each involves the freezing or crystallization of a substance in the pores or cracks, which results in a heavy interior strain.

State highway officials are recognizing the need of more uniform specifications, and recently have instituted a set of tentative standards[4] for material to be used in macadam and concrete highways.

Also, much study is being devoted to the proper sizing of aggregate and the proper proportioning of the various sizes to obtain maximum strength and durability with a minimum of cement. It is desirable to use a combination of different sizes that will give the lowest percentage of voids.

4 Tentative Standard Specifications for Highway Materials of the American Association of State Highway Officials: Washington, 1929, 56 pp.

Road Stone.- Stone of various sizes is used in the construction of bituminous and macadam roads. Material under one fourth inch is classed as fine screenings and is used principally for waterbound macadam. Coarser screenings, up to one half inch, serve as fine aggregate for bituminous concrete. Sizes between one fourth and three fourths inch are dustless screenings or chips for surface treatment of bituminous roads. Coarse chips, between three-fourths and $1\frac{1}{2}$ inches, are used for bituminous macadam. Sizes from $1\frac{1}{4}$ to $2\frac{1}{2}$ inches are suitable for the wearing course of both waterbound or bituminous macadam. Rock between $2\frac{1}{2}$ and $3\frac{1}{2}$ inches constitutes the base course of highways.

The requisite qualities of road stone are similar to those of aggregate except that resistance to abrasion is of primary importance. A soft rock disintegrates under traffic, and a laminated rock, even if fairly hard, breaks into flat or elongated pieces which will not compact solidly. Rock of low porosity is desirable, as otherwise water may penetrate and soften the structure of the road. Road stone should break into sharply angular, chunky fragments which when properly graded as to size will interlock and press firmly into the surface of the road.

Various standard methods of testing, sampling, and mechanically analyzing road materials are given in the American Society for Testing Materials Standards, 1927, Part 2.

Railroad Ballast.- Many thousand tons of limestone are consumed by railroad companies for the maintenance of their road beds. A minimum of three fourths inch and a maximum of $2\frac{1}{2}$ inches for ballast sizes has been fixed. The requirements as to quality are generally the same as for aggregate and road stone.

Asphalt Filler.- Limestone dust, approximately 80 percent of which will pass a 200-mesh screen, is used as a filler in road asphalt-surface mixtures. Many thousand tons are consumed annually in this way. For the most part it is regarded as a by-product outlet, though the preparation of asphalt filler is an appreciable part of the business of some quarrying companies.

Riprap.- Riprap consists of heavy, irregular rock used chiefly for river and harbor work, such as spillways at dams, shore protection, and docks. It is a low-priced product and is usually procured from near-by quarries. Any type of dense, sound limestone may be used, there being no general specifications.

Coal-Mine Dusting.- Several hundred operators of bituminous coal mines now employ fine incombustible dusts for distribution throughout their mines as a means of preventing or checking coal-dust explosions. Limestone dust is particularly satisfactory for this purpose because it is essentially carbonate of lime, which is not injurious to the lungs, and because being white the proportion of inert material present can be readily estimated. However, limestone with high silica content is not desirable for dusting. Screen-size

specifications are not exacting. Those approved by this Bureau require that 100 percent shall pass through a 20-mesh screen, and 50 percent through a 200-mesh screen.

As the material commands a low price, it is usually furnished by quarries situated near the coal mines. Approximately 60,000 to 70,000 tons of pulverized limestone are consumed annually in this way.

Sewage Filter Beds.- Crushed limestone has been found satisfactory as sewage filter beds. High-calcium or dolomitic limestone may be used, and siliceous impurities are not objectionable if they are fine grained and evenly distributed. Certain other impurities, notably pyrite, marcasite, and clay, are to be avoided. The rock should be strong and compact with pore space evenly distributed, and the fragments should be sufficiently rough to provide anchorage for bacteria. Fines and dirt should be screened out.

Stucco and Terrazzo.- Dense compact limestones of attractive colors may be crushed into small fragments for terrazzo floors. Similar material reasonably impervious to moisture finds some use in stucco and pebble-dash work.

Poultry Grit.- Limestone crushed to granular form and screened to uniform sizes is sold for poultry grit. Almost any type of limestone may be so used.

Sand.- Limestone crushed to the size of sand grains when carefully washed and graded may be substituted for silica sand in mortar, wall plaster, and concrete. Mortar tests as reported by Kreige[5] showed strengths considerably in excess of those obtained with standard silica sands. A large tonnage of limestone sand has also been used in concrete highway construction in the Middle West.

Roofing Gravel.- Screened limestone chips are sold as gravel to be used with tar for coating flat roofs.

Yard and Playground Surfacing.- Screenings without a binder afford good surfaces for school yards, playgrounds, walkways, station platforms, and tennis courts.

Concrete-Block Manufacture.- Limestone screenings are used as aggregate in the manufacture of concrete blocks, and limestone chips may be embedded in the surface to make the blocks resemble cut stone.

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5 Kreige, Herbert F., Washed Limestone Sand: Pit and Quarry, vol. 17, No. 11, 1929, pp. 64-66.

Chalk, Whiting, and Whiting Substitute.- Chalk is a noncrystalline, soft, friable, fine-grained, light-colored type of limestone. The distinguishing physical characteristics of true chalk never have been fully defined; probably its noncrystalline and colloidal properties are most important. Whiting is a pulverized, purified, and carefully sized chalk. Whiting substitutes consist of finely ground limestone or dolomite, ground marble, white marl, and chemically precipitated calcium carbonate. Very little true chalk has been produced in the United States.

True whiting is preferred for calcimine and cold-water paints and in the manufacture of putty. Other uses of whiting are as a ceramic raw material and as a filler in numerous products, such as rubber, paint, paper, oilcloth, window shades, and linoleum. The uses are discussed in more detail in another publication of the Bureau.[6]

Uses for Which Chemical Properties are Most Important

The importance of the chemical composition of limestone for certain uses is not always to be interpreted as a demand for pure calcium carbonate. The manufacture of cement, for example, calls primarily for a proper balance between the chemical constituents, and suitable limestone may be quite impure. Chemical purity, however, is demanded for some purposes.

Cement Manufacture.- Limestone is the chief raw material of cement. While pure limestone is not required, constancy in chemical composition is desired. The general requirements are that (1) rock should be free of concretions of iron mineral; (2) the silica and alumina contents should be sufficiently low and in such proportions that they will not interfere with the desired silica-alumina ratio in the finished product; (3) the magnesium content should be low enough that the finished product will not contain more than 5 percent magnesia (MgO); (4) the content of iron should be sufficiently low that the ferric oxide content of the cement does not exceed 4 percent; (5) the sulphur content should be low.

Cement rock is simply an argillaceous limestone which contains enough clay as it occurs in nature to adapt it for the manufacture of cement. Sometimes it may be necessary to adjust its composition by adding small amounts of either high-calcium limestone or clay. The cement industry consumes many millions of tons of limestone every year, but the successful operation of a cement plant depends upon fuel supplies, transportation facilities, market demands, production costs, and competitive conditions, as well as upon suitable raw materials.

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6 Bowles, Oliver, Chalk, Whiting, and Whiting Substitute: Inf. Circ. 6482, Bureau of Mines, 1931, 13 pp.

Lime Manufacture.- Between 8,500,000 and 9,000,000 tons of limestone are used annually in the United States for lime manufacture. For such use stone must conform with rather rigid physical and chemical requirements. Exceptionally pure stone, total carbonates ranging from 97 to 99 percent, is generally used because practically all impurities remain in the lime that results from calcination. The most common impurities are silica, alumina, iron, and sulphur. The stone should be sound and compact because porous and friable types break down during calcination, and the large quantity of fines resulting from quarrying can not be used in shaft kilns. Both high-calcium and high-magnesium limestones, or dolomites, are used.

Lime enters three important fields of utilization - the chemical industries, the building industries, and agriculture. High-calcium limes are used chiefly for mortars and chemical purposes, and magnesian limes for finishing plasters. Both high-calcium and high-magnesium limes are used for soil treatment, and for this use a larger percentage of impurity is tolerated than for most of the other applications.

Furnace Flux.- Normally from 20,000,000 to 24,000,000 tons of limestone are used annually as fluxing material in the iron furnaces of the United States. As the chief purpose of flux is to remove silica and alumina from ores, it is evident that limestone employed for this purpose should have a small content of these elements. If an impure stone is used, part of the carbonate content is absorbed in fluxing off the foreign elements in the stone, which reduces the amount available for removing impurities of the ore. "Available carbonate" is a term applied to the percentage of calcium and magnesium carbonates available for fluxing ore after a sufficient percentage has been deducted to neutralize impurities in the stone itself. While a low content of impurities is desirable, conditions greatly influence the use of pure and impure stone. Silica and alumina in a fluxing stone do no real harm in a blast furnace; they merely make the stone less effective, increase slag volume and fuel consumption, and slow up production to a limited extent. If the price differential between an impure stone and one of high chemical purity is sufficient to offset these disadvantages, the lower-priced product may be more economical to use. Generally the presence of magnesium in the stone is not regarded as objectionable. Sulphur in fluxing stone should not exceed 0.5 percent. The upper limit of phosphorus for Bessemer iron is placed at 0.01 percent and for non-Bessemer iron at 0.1 percent.

A relatively small tonnage of limestone is used in the manufacture of basic open-hearth steel. Purer stone is required for making steel than for blast-furnace work; the silica content is usually limited to 1 percent, and the alumina content to 1.5 percent. As one of the chief purposes is to remove phosphorus, and as magnesium is a poor phosphorus remover, the maximum permissible content of MgO is usually 5 percent. More complete information on both production and use of fluxing stone is given in a Bureau report.[7]

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7 Bowles, Oliver, Metallurgical Limestone: Bull. 299, Bureau of Mines, 1929, 40 pp.

Agricultural Uses.- Limestone finds important applications in agriculture as a fertilizer, soil conditioner, and as a corrective for soil acidity. Pure limestone is desirable though not essential, for while impurities lessen the percentage of calcium or magnesium available for improving the soil, they are not injurious to plant growth. Therefore, local impure limestone may be more economical than higher-grade material that must bear a relatively heavy transportation charge. There is some difference of opinion regarding the suitability of dolomitic limestone, but most authorities agree that magnesium is of equal value with calcium and that the value of stone for agricultural purposes may be measured by the percentage of total carbonates present.

A small amount of limestone is used with commercial fertilizers as a diluting material or filler.

Stock Food.- Pulverized limestone is added to stock food as a bone builder.

Alkali Manufacture.- The manufacture of sodium carbonate (soda ash) consumes from 4,000,000 to 5,000,000 tons of limestone a year. The Leblanc process (almost obsolete) involves a reaction between limestone, sodium sulphate, and carbon to form the desired sodium carbonate. A furnace charge consists of 100 pounds of salt cake (sodium sulphate), 100 pounds of limestone, and 50 pounds of coal-dust. The magnesia and silica content of the limestone should be low, or loss will ensue through the formation of insoluble residues.

For the ammonia or Solvay process, which is generally used, the principal reaction is in a brine. Carbon dioxide and lime are obtained by calcination of limestone in special continuous kilns. Chemically pure limestone, though not essential, is desirable. About $1\frac{1}{4}$ tons of limestone, preferably in pieces 1 to 6 inches in size, is used per ton of soda ash produced.

Calcium Carbide.- Calcium carbide is manufactured by fusing an intimate mixture of powdered lime or limestone and coke in an electric furnace. About 2 tons of limestone is required for each ton of carbide made. Very pure stone is required. The content of phosphorus should be less than 0.01 percent; magnesia, less than 2 percent; and silica, less than 3 percent.

Sugar.- Limestone is used in large quantities in refining beet sugar. It is calcined into lime and used in milk-of-lime suspensions to precipitate impurities from the juices, or, in the Steffen process, to precipitate sugar from impure solutions. Calcination is preferably done at the refinery as the carbon dioxide is recovered for use in subsequent treatment of juices. The lime requirement is about $2\frac{1}{2}$ percent of the weight of beets.

Limestone generally used is in sizes from 2 to 6 inches and averages 97 to 99 percent calcium carbonate. Silica is detrimental, and iron oxide affects the color of the sugar. According to specifications[8] limestone (calcined before analysis) for the Steffen process should contain at least 90 percent of sugar-soluble lime and not more than 3 percent of magnesium oxide.

Glass.- Either lime or limestone is used in the manufacture of glass. Limestone of uniform grade is required because of the rigid control necessary in composition of the batch. The iron content should be low because of its coloring effect; for optical glass it must be practically zero. Silica in moderate amounts is not detrimental, but the sulphur and phosphorus content should be low. Combined calcium and magnesium oxide requirements (on a calcined basis) are about as follows: At least 89 percent for bottle glass, 91 percent for sheet glass, 93 percent for blown glass, 96 percent for rolled glass, and 99 percent for optical glass.

Rubber.- Limestone or its products are used in two ways in rubber manufacture - as whiting and as hydrated lime. The whiting acts as a filler and also assists in controlling hardness and elasticity in building compounds. Powdered chalk is used in the manufacture of rubber cement.

Paper.- The acid liquor used in one method of manufacturing paper is obtained by treating either milk of lime or wet limestone with sulphur dioxide. Dolomite or high-magnesian limestone is preferred for preparing the lime, and should be pure enough to give a product containing not more than 3 percent total iron oxide, alumina, silica, or other insoluble impurities.

Another method of obtaining acid liquor is by the Jennsen tower system, whereby sulphur gases pass through a tower packed with limestone. Stone for this purpose should have not more than 2½ percent of magnesium carbonates or other impurities, and a calcium carbonate content of at least 95 percent is recommended. Medium-grained stone in fragments from 8 to 14 inches is used.

Carbon Dioxide.- During recent years the use of carbon dioxide has increased greatly, chiefly because of its employment in solid form as a refrigerant. Dolomite is used to some extent as a source.

Mineral Wool.- The name "mineral wool" or "rock wool" is applied to fine interlaced threads of calcium silicate used chiefly in heat insulation. It is made by melting argillaceous limestone in a cupola furnace and blowing the slag which is formed into fine threads by means of a steam jet.

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8 U. S. Bureau of Standards, Recommended Specifications for Limestone, Quicklime, Lime Powder, and Hydrated Lime for Use in the Manufacture of Sugar: Circular 206, 1925, 6 pp.

Uses of Dolomite and High-Magnesian Limestone

Some of the uses for which a magnesium content is essential or preferred are covered in preceding paragraphs. There are in addition several special uses.

Refractories.- Dolomite and high-magnesian limestone are used extensively as refractory linings in metallurgical furnaces. The dolomite may be dead-burned or used in the raw state. A dead-burned product is made by calcining the raw material at a temperature of about 1500° C.

Technical Carbonate.- Dolomite is used in large quantities for the manufacture of basic magnesium carbonate, which is employed chiefly for pipe and boiler covering.

Further details on the uses of dolomite are given in a recent Bureau report.[9]

Quarry Methods

Prospecting.- Development work should not be started on a deposit without reasonable assurance of an available mass of rock sufficiently high in quality and abundant in supply for profitable exploitation. If the rock appears in bare outcrop, usually a rough estimate as to its quality and extent can easily be made. Limestones are as a rule fairly constant in composition throughout the same bed or zone of deposition; the greatest variations are found in passing from one bed to another. Therefore, all beds that may be included in a quarry are usually sampled. If a cross section is not available in nature, test holes are drilled at such intervals as will supply adequate data covering the whole area under consideration. Churn drills are usually employed, and samples of the cuttings for analysis are taken at regular intervals. No definite rules can be given either for the position or arrangement of holes. In flat-lying beds of uniform thickness and fairly constant composition they may be spaced at wide intervals - 100,500, or even 1,000 feet; where beds are folded or tilted or where changes in composition or structure occur within short distances, they should be more closely spaced. Accurate permanent records of all drilling are highly desirable. The direct cost of sinking churn-drill holes $5\frac{1}{2}$ to 6 inches in diameter in limestone varies from 20 to 60 cents a foot.

When the extent of a deposit is known the approximate tonnage may easily be determined. Limestone weighs on an average about 160 pounds a cubic foot. To determine the approximate number of short tons available in a deposit, the length, width, and depth in feet may be multiplied; this product is multiplied by the average weight per cubic foot, 160 pounds, and divided by 2,000.

9 Hatmaker, Paul, Utilization of Dolomite and High-Magnesium Limestone: Inf. Circ. 6524, Bureau of Mines, 1931, 18 pp.

More complete data on quarry methods are given in other reports of the Bureau.[10]

Generally it is deemed unwise to expend money on quarries, crushers, and screening plants unless a reserve of good rock sufficient for at least 20 years' operation is assured.

Stripping.- Stripping is the process of removing the overburden of clay, gravel, or sand from the rock surface. The depth of overburden varies from a few inches to 10, 20, 30, or even 40 or more feet. Likewise, the nature of the materials composing it is variable. It may be easily disintegrated loam, sticky plastic clay, sand, gravel, boulders, or a hardpan that may require blasting. The presence of erosion cavities in the limestone surface may make stripping difficult. Many different stripping processes are employed. The more important of them are the hydraulic method, the drag-line scraper, power shovel, clamshell bucket worked from a derrick arm, scrapers hauled by horses or mules, or hand methods with picks and shovels. Costs of power-shovel stripping, the most common method, vary under average conditions from 30 to 50 cents a cubic yard. A detailed discussion of stripping is given in a former report.[11]

Plan.- The method of quarrying a deposit depends on thickness and dip of beds. If the strata are horizontal or inclined at low angles, open-pit quarries are developed except where the overburden is so heavy that underground work is better. When such beds are thin, the pit must be enlarged laterally; when thick, deeper and narrower quarries are made.

When beds are tilted at steep angles and are many feet thick, an open quarry may be worked to considerable depth, but removal of waste to avoid a dangerous overhang involves ever-increasing expense as the floor is deepened. When tilted beds are thin, any lateral extension must be in the direction of the strike or outcrop. The narrowness of the working face of a thin bed cramps operations and makes it difficult to obtain large daily tonnages, but quarrying at several levels partly overcomes this condition. Underground methods are commonly followed where the overburden is excessive or where comparatively narrow beds stand at steep angles.

Drilling.- The churn drill, commonly called the "well drill," is the type most widely used for primary drilling, although piston drills are also used for this purpose. Hand-manipulated compressed-air hammer drills are so employed to some extent, but they are used chiefly in secondary drilling in preparing pop shots to break up larger fragments.

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10 Bowles, Oliver, Rock Quarrying for Cement Manufacture: Bull. 160, Bureau of Mines, 1918, 160 pp.
Bowles, Oliver, and Myers, W. M., Quarry Problems in the Lime Industry: Bull. 269, Bureau of Mines, 1927, 97 pp.

11 Bowles, Oliver, and Myers, W. M., Quarry Problems in the Lime Industry: Bull. 269, Bureau of Mines, 1927, pp. 15-26.

As the purpose of drilling is to obtain space for explosives, the only fair method of comparing costs is to consider drilling not in terms of cost per foot but rather on the basis of the volume of the hole obtained. Churn drills are usually preferred because under average conditions when compared with the smaller drills they provide larger space for explosives at lower cost. The advantage probably lies with small drills for shallow benches and with churn drills for high benches.

A comparatively large space for explosives is sometimes provided by "springing" small drill holes. The process consists of discharging in the bottom of the hole a succession of small explosive charges, thereby pulverizing and removing the rock to form a cavity.

Blasting.- Heavy blasting to break rock from the parent ledge is known as primary blasting. When a primary shot is not sufficient to break the rock to sizes small enough for loading, secondary blasting may be employed. It is generally considered that heavy blasts in churn-drill holes are the most effective, but advocates of small-hole blasting think that a more general distribution of the explosive throughout the rock mass breaks it more completely, and it requires less block-hole shooting than does churn-drill blasting. Undoubtedly different results are obtained in different types of rock. Blasting in deep churn-drill holes is the method commonly employed. The charge in each hole should be regulated according to the estimated tonnage of rock to be moved. In average practice a pound of 40 percent ammonia dynamite shatters 3 to 6 tons of rock, the amount depending on the toughness of the rock.

For practically all primary shots explosives are fired simultaneously. When electric detonators are used the drill holes are connected either in series or in parallel or multiple. Another way of firing is to use a detonating fuse consisting of a lead tube filled with trinitrotoluene. A main line connecting the branches from each hole is attached to the detonator, and all charges are fired at practically the same time.

A method adaptable for blasting a high face when the strata are irregular is to fire charges in small tunnels driven into the quarry face at the floor.

To promote blasting efficiency many quarry superintendents keep accurate records which show for each shot the number and depth of holes, spacing, burden, kind and weight of explosive in each hole, tonnage of rock moved, and condition of fragmentation.

There are two common methods of secondary blasting. "Mud-capping" consists of placing a stick of dynamite with attached fuse on the surface of the rock to be broken and covering the dynamite with clay which confines the explosion and directs it toward the rock. "Blockholing" is much more

effective. Holes several inches deep are drilled, and a stick, or part of a stick, of dynamite with a fuse attached is placed in each one. Rock-dust or clay is used for stemming. A number of blasts are discharged in rapid succession.

Ammonia dynamite is most commonly used in quarry work. Gelatin dynamite should be used in wet holes. Liquid oxygen (commonly designated "L.O.X.") used to some extent as a substitute for dynamite is quite effective, but its use is rarely advised except where quarries are located near a liquid-oxygen manufacturing plant or companies are large enough to justify the manufacture of their own supplies.

Loading.- Loading rock into cars is the largest single item of quarry expense. It is done by hand at many quarries supplying lime plants or producing flux for furnaces because pure stone has to be provided and this method affords a means of selective loading with rejection of siliceous or otherwise impure fragments.

In quarries producing aggregate, road stone, or ballast, power shovels are generally used. For a daily output of 150 to 300 tons of rock small tractor shovels with $\frac{3}{4}$- to $1\frac{1}{4}$-yard dippers are suitable, but shovels with dippers capable of handling 5 to 10 tons are employed in the larger quarries. Records obtained from a number of quarries a few years ago show an average daily output of 112 tons per man (pitmen and shovelmen only) by power shovel as contrasted with 16 tons per hand-loader. Very little secondary blasting is necessary when power shovels load the rock, whereas not only blockholing but a great deal of laborious sledging must be done when it is removed by hand. However, the mechanical method requires expensive accessory crushing and screening equipment, and the large investment thus involved may make it more profitable to load by hand at small quarries.

Haulage.- Haulage involves the motive power and equipment required to convey rock from the loading place to some point outside the quarry where it is transshipped, crushed, or otherwise treated.

The arrangement of tracks depends on the loading method and the size and shape of the quarry opening. When loading is done by hand it is desirable to have many working places, each with independent trackage from the main line. For power-shovel loading two systems are followed. When the face is wide, the track parallels it, and a succession of cars are loaded; when the face is narrow, the track runs directly toward it and ends in a Y. Tracks from pit quarries may be inclined; many are so steep that cables are used. Tracks from shelf quarries may be level or moderately graded. The gage of tracks ranges from 24 to 42 inches. Low cars of 2- to $2\frac{1}{2}$-ton capacity are popular in small quarries, but larger and stronger cars are used for power-shovel loading. Side-dump cars are most commonly used.

Various types of motive power are used. Careful adjustment of grades makes car movement largely automatic. On a gentle grade loaded cars may proceed by gravity from the face; the empties may be returned by horses or mules. Animal power has been preferred for short hauls, but smaller types of electric or gasoline locomotives are replacing it. When the haulage distance exceeds 500 feet, locomotives are generally employed, and they may haul as many as 20 cars. Cable and drum haulage is commonly employed on inclines. Trucks are another means of conveyance.

Underground Mining

The chief factors that make underground mining of limestone advantageous at times are (1) a heavy overburden of soil or inferior rock blanketing a flat-lying deposit of good stone; (2) an inclination of beds of serviceable material that demands too great an extension of the pit along the strike or outcrop and results in an increasing overburden as the pit is enlarged in the direction of the dip; and (3) the necessity for working at increasing depths as surface deposits are exhausted. Crushed limestone is too low-priced a product to justify the expense of timbering mines except possibly in shafts or entries; hence, mining is successful only when the rock is strong and massive enough to constitute the roofs and supporting pillars.

The principal advantages of underground work are avoidance of stripping expense and soil contamination, and protection of laborers from inclement weather. However, drilling and blasting are more expensive than in open-pit work, the proportion of fines is increased, and from 20 to 25 percent of the stone must be left as pillars. In making a choice between underground and open-pit methods each operation must be considered on its own merits.

Mining methods are fully described by Thoenen in a publication of this Bureau.[12]

Preparation of Materials

Crushing.- Four main types of crushers are in general use - the gyratory crusher, the jaw crusher, double rolls, and single rolls; cone and disk crushers are also used. The size of a crusher should be governed by the size of the blocks to be crushed and by the daily output desired. The first cost and power charge of large crushers are high, but maintenance expense is usually low, and no delays are caused by jammed blocks.

Screening.- Crushed stone is assorted according to size by some type of screen. For separation of the larger sizes an inclined railroad rail or bar grizzly with 3½- to 5-inch spacing is sometimes used. The rotary screen or

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12 Thoenen, J. R., Underground Limestone Mining: Bull. 262, Bureau of Mines, 1926, pp. 33-80.

trommel has been most widely employed. More recently, however, for coarser sizing or scalping, rotary disks such as the cataract grizzly or the multiroll sizer have been adopted. The advantages claimed are long wear, absence of vibration, and minimum grinding and breaking, as there is no cascading action. The finer sizes are usually graded by vibrating screens.

Washing.- Demand for clean stone with a minimum of fines has led to the addition of washing equipment at many quarries. Washing is particularly desirable at quarries where clay seams occur. It is generally accomplished by directing water on the stone as it cascades in a trommel or passes over rotary disks or vibrating screens.

Elevating and Conveying.- When a crushing plant is located on the quarry floor, the stone may be elevated to screens or to storage by bucket elevators or belt conveyors. Cascading from high elevations is to be avoided because it produces fines, particularly if the limestone is soft.

Fine Grinding.- Limestone is reduced to ground form usually by dry-grinding processes. When prepared in this manner the moisture content must be low, and hence the stone is usually passed through rotary dryers. Roller mills of various types, also impact, beater, or swinging-hammer mills are used. For very fine-grained products ball or pebble mills are preferable.

Classification of the smaller sizes is accomplished by air separation, and of the coarser material by vibrating screens.

Grinding efficiency is increased by the removal of material from the system as soon as it is ground. In closed-circuit grinding the coarser particles are returned to the system for further reduction. Wet processes are sometimes preferred for the preparation of whiting or marble flour. Size classification is then accomplished by water settlement.

Operating Costs

Average costs are difficult to estimate because conditions are so variable in different quarries that both individual items and total costs are quite diverse. Thoenen[13] estimated a cost of 67 cents a ton as an average for 30 open-pit limestone quarries in various parts of the country. For an average quarry operating on a large scale this total might be distributed as follows: Stripping, 6; drilling, 9.5; explosives, 7.5; loading (hand), 22; mucking, 6; haulage, 5; repairs, taxes, etc., 5.5; interest and amortization, 5.5. If a power shovel is used, the direct loading cost would be much less than 22 cents, but with interest on investment, together with additional crushing expense, the total would probably differ little from the hand-loading cost. A cost analysis by Thoenen[14] of 110 limestone quarries grouped according to size and equipment shows direct quarrying and crushing costs ranging from 35 to 95 cents a ton.

13 Thoenen, J. R., Work cited, p. 94.
14 Thoenen, J. R., Study of Quarry Costs: Rept. of Investigations 2911, Bureau of Mines, 1929, p. 2.

Markets

Crushed stone is a low-priced product and as the transportation charge is usually a large part of the delivered price, it has a relatively limited market range. Profitable operation depends largely on the extent of local markets. Producers are most concerned about steady market requirements of near-by builders, contractors, State highway departments and other users. A wise operator gages his plant capacity by the normal demand but is prepared to profit by any extraordinary market opportunities.

Demand is influenced by availability and cost of other materials and by competition of concrete with brick, stone, or other building products.

Prices

Prices of crushed limestone are subject to local conditions of production, cost, and competition and vary widely even within restricted areas; therefore, the determination of selling price is a problem for each producer. About 50 quotations representing the chief centers usually appear in the market columns. The average value of the 1931 production as reported to the Bureau of Mines was 92 cents a ton f.o.b. plant - the lowest average since the World War. The highest average figure during the same period was $1.19 a ton in 1920.

Royalties

Many crushed-stone producers operate in deposits which they do not own. It is customary in such instances to pay the owner of the property a royalty of so much per ton of crushed stone sold. Royalties vary from 1 to 10 cents a ton depending on local conditions. The lower figures usually prevail where production is large, but sales value per ton, production cost, or competition may influence the amount. A minimum average daily or monthly production is usually a condition of a royalty agreement.

Capital Requirements

A prospective operator desires to know how much capical he must have in order to establish a crushed-stone industry. The most reasonable basis for expressing investment is the capital required per annual ton of production. Thus, if $1,000,000 is needed to build a plant capable of producing 1,000,000 tons a year, the investment is $1 an annual ton. Capital investment, like costs, varies greatly from plant to plant, but average figures are at least indicateive of what one may expect.

A detailed study[15] of 64 crushed-stone plants in the United States shows an average capital investment of $1.25 an annual ton of average production over a 2-year period. This figure is based on depleted values representing actual replacement values of the properties. Therefore, a prospective producer who is just beginning operation must estimate his initial investment at a somewhat higher rate than the figure given above. Of the total capital requirement land and mineral constitutes about 15 percent; and plant and equipment, about 85 percent.

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15 Bowles, Oliver, Economics of Crushed-Stone Production: Econ. Paper 12, Bureau of Mines, 1931, p. 53.

I. C. 6724

June, 1933

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

PROTECTIVE CLOTHING IN THE MINING INDUSTRY

BY

R. D. CURRIE AND W. J. FENE

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

PROTECTIVE CLOTHING IN THE MINING INDUSTRY[1]

By R. D. Currie[2] and W. J. Fene[2]

The value of protective clothing in the prevention of injuries has long been recognized in many of the leading industries, but the adoption of the idea into mine safety programs is comparatively new. Once started, however, the idea has grown rapidly and the results already obtained by mining companies have shown that it is fundamentally sound.

Both coal and metal mining companies are rapidly adopting these protective measures, and since 1924 many mines have become well equipped with eye, head, and foot protection. At the present time approximately 70 percent of the employees in the anthracite mines of Pennsylvania are provided with safety hats. At many metal mines all employees are provided with safety hats, shoes, and goggles. The Pennsylvania mining law provides that all shaft workers in anthracite mines shall wear protective hats.

According to a study[3] made of the nature of 3,485 temporary injuries and of the parts of the body of underground coal mine employees most exposed to occupational hazards, the head and face were injured 665 times, the hands 856 times, and the feet 565 times. Of 2,555 temporary injuries in metal mines, the head and face received 484, the hands 788, and the feet 357. In a similar study[4] made by a coal-mining company for the year 1928, of 556 injuries the scalp received 9, the skull 4, the eyes 47, the hands 62, the thumbs 21, the fingers 141, the ankle 7, the feet 51, and the toes 35. Probably most of these injuries could have been prevented by suitable protective clothing and equipment. This study also revealed that at least one injury occurred in each of the 34 classifications of workers employed at these mines.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6724."
2 Associate mining engineers, safety division, U. S. Bureau of Mines, Pittsburgh Experiment Station, Pittsburgh, Pa.
3 Adams, W. W., Mine Accident Statistics: Rept. of Investigations 2641, Bureau of Mines, September, 1924, pp. 26, 27, 49-50.
4 Currie, R. D., Coal Mine Safety Organizations in Alabama: Technical Paper 489, Bureau of Mines, 1931, p. 34.

In addition to being most numerous, injuries to the parts of the body previously named are the most easily prevented by suitable clothing and equipment. Therefore the use of protective clothing is becoming popular with those companies who have made a study of its advantages.

EYE PROTECTION

Eye protection was probably the first safety need to be considered, and, as was natural, the type of protective goggles first adopted by the mining industry were of the type in use in other industries. As might be expected it was soon found that goggles suitable for the mill or shop were not necessarily suitable for mining. Many complaints were received from the men that the goggles fogged, obscured the vision, or were too heavy for continuous wear. These complaints usually were well founded, and considerable work had to be done in developing goggles suitable for the mining industry. To be satisfactory, goggles should be light in weight, well ventilated, rugged, optically true, and should allow a wide angle of vision, if their voluntary use by the miner is desirable. Strict discipline might cause employees to wear goggles with undesirable characteristics, but eye injuries are likely to continue to occur where such compulsory measures are necessary.

Some objections have been made to the use of goggles by underground employees on the grounds that the wearing of goggles would expose the workmen to other hazards and indirectly bring about more serious injuries from other causes. This theory has not been substantiated, as is shown by one large coal company which has made a close study of the factors entering into the use of goggles and whose experience over a period of 4 years has been that as far as could be ascertained not a single injury from other causes was attributable directly or indirectly to the wearing of goggles.

There are many types of safety goggles on the market, but the rigid type, provided with shatterproof lenses, appears to be best suited for underground workers and is the type most generally worn. In some cases corrective lenses have been provided for those having defective vision; this very good practice should unquestionably be extended and ultimately be universal. While full vision for underground workers is probably even more necessary than for workers in surface industries, there are few workers who wear ordinary eyeglasses underground to correct defective vision. It has been found that a large percentage of industrial workers have defective vision, and there is no doubt that the providing of corrective lenses in goggles would not only protect the eyes from flying particles, but due to the better vision provided, would also help to prevent accidents from other causes. To obtain the maximum benefit from the use of goggles, they should be worn during the entire shift, or from the time the workmen enter the mine until they leave; this is seldom required and is done only in rare instances, but there is no question that it is advisable. Goggles also should be worn by surface employees when engaged in any work where particles are liable to fly into the eyes, such as grinding, chiseling, or hammering iron or steel.

HEAD PROTECTION

Head and scalp protection is a problem that probably concerns the miner more than any other worker, and for this reason it is only natural that the development of head protection has been almost exclusively for the mining industry. The first protective headgear was, undoubtedly, a padding of cotton waste stuffed into a miner's cap or felt hat. The present protective hats and caps are such a vast improvement over former makeshift headgear that the mining industry is adopting the new type with considerable rapidity. The first step in the development of a protective hat in metal mining was to use a steel hat or helmet like that which protected the soldiers in the Great War, and the first in coal mining was to stuff the hat with cotton waste. Later developments, however, have provided a hard hat with a cushioned lining to absorb the force of a direct blow and allow ventilation, which is satisfactory for the use of both coal and metal miners.

The types of protective hats and caps now in use are constructed of hard insulating material that will withstand a hard blow or a high-voltage electric current; they are moisture- and acid-proof and are made in various styles to meet various mining needs. They will prevent many injuries that might otherwise prove serious or fatal. It is important that the wearer procure the proper size for his head and that the cushion be properly adjusted.

Objections were made to some of the first protective hats introduced for the reason that they were heavy and uncomfortable to wear. These objections have been overcome and the hats now manufactured are lighter in weight, better ventilated, and are worn with little or no discomfort.

The experience of many companies is that protective hats are preventing fatal as well as less serious accidents. One company is now mining four times as much coal per head injury as it did before it introduced protective hats. Another company reduced its compensation and medical cost due to head injuries approximately $7,500 in 4 years' time by the use of protective hats. It is generally found that as the number of protective hats increase at a mine the number of head injuries decrease.

TOE PROTECTION

Toe protection has been difficult to arrange, primarily because miners, in general, have been in the habit of wearing shoes for work that are no longer suitable for street wear, and are reluctant to purchase special shoes for work. Another drawback is the fact that many of the protective shoes introduced into the industry are not properly constructed to give comfort to the wearer; this is not so noticeable in work shoes lacking the stiff box toes, but is accentuated when stiffness is added. Another drawback to the voluntary adoption of foot protection is the limitation in the selection of lasts offered by the shoe manufacturer; in general, only wide-last shoes are available to the prospective purchaser of safety shoes, and it is improbable that more than half the men in the industry can be properly fitted in the

shoes now available. This, however, is a difficulty that can be easily remedied through proper cooperation between mining companies and shoe manufacturers. Proper designs in rubber footwear having protective toes have apparently given more difficulty than leather footwear, but most of these difficulties are also caused by the lack of selection in sizes and widths.

Due to the nature of the material handled and the increasing use of machinery, the need for foot protection for underground employees is urgent. Protective shoes have prevented many toe injuries due to falling or rolling coal, rock, ore, and timbers, and from wheels of moving cars and machines, and their use in and around mines should be universal.

OTHER TYPES OF PROTECTIVE CLOTHING

Other types of protective clothing that are required by some companies are gloves, leggings or high-cut boots, and safety belts.

A study of the percentage of accidents happening to fingers and hands indicates that some type of suitable protection should be given to these accident-prone parts of the body. There have been so many accidents, however, traceable to the use of certain types of gloves and to the injudicious use of gloves near fast-moving machinery, that great care must be exercised in adopting this mode of protection to prevent hand and finger accidents. The suitable type of glove when properly used should, however, prevent many small hand and finger injuries. Infections from small cuts have been greatly reduced in mines where the use of gloves is required.

Leggings, high-cut boots, or snug-fitting trouser legs undoubtedly afford real protection to mining employees who are constantly working in close proximity to moving cars, locomotives, conveyors, and other machinery and equipment. Employees should not be permitted to endanger themselves by wearing loose or baggy clothing, ragged sleeves, or loose neckties.

Safety belts are needed only in certain types of mining and under certain conditions of work, so that they will never be used generally; their use, however, where the conditions require them, as in glory holes, steep banks in opencuts, and similar places, is a real safety factor that should not be overlooked.

In addition to these widely used items of protective clothing, certain companies require employees working in unusually hazardous occupations to wear special protective clothing. Wiremen in many mines are required to wear rubber gloves, rubber shoes, or both while working on power and trolley lines. Haulage crews in some Alabama coal mines are required to wear shoes having rubber inner soles to insulate against electric current, while at least one company requires the use of a foot guard to prevent injuries to the feet of some of their tipple crew. Some companies that load coal direct from the tipple into river barges require the barge tenders to wear life belts.

COST AND DISTRIBUTION

The initial cost of installing protective clothing is an important item and is generally a controlling factor in the selection of types of protective clothing adopted or a deciding element in whether such protection is adopted.

Other questions of cost and distribution which confront every company considering the adoption of such protective devices are:

1. Who should pay for this protection, the company or the employees?

2. How should distribution be made?

3. How can the company expect a fair return for its money if it furnishes these protective devices?

4. How can the miner expect value for money paid if he is required to purchase them?

5. Should the use of such equipment be voluntary with the workmen or made compulsory?

1. Expense. - The first of these questions is perhaps the most difficult to answer and is often the criterion as to the success or failure of the protective-clothing program. Some companies have answered it by furnishing certain items of protective clothing while requiring the employees to purchase the others. Recent legislation in some mining States has made it compulsory for certain employees to be equipped with protective clothing, while the compensation rating schedules of other mining States have made it worth while to have all employees equipped with certain items of protective clothing. In any event, both the employer and employee are protected by the general use of these devices, and it seems only reasonable that both should share in their initial cost.

2. Distribution. - The second question is one that should cause little difficulty because most mining companies already have a distribution system for general mine supplies and needs for their employees, so that little difficulty should be encountered in adding these items to their stock. There are many advantages in this form of handling these items; the most important one is the control afforded the company in checking the type of equipment supplied to their men, and secondly, the assurance of a fair price for items purchased by the men.

3. Value to Company. - The answer to the third question depends upon several factors. The savings made in compensation and medical cost will usually more than cover the cost of such equipment. A recent report from a coal-mining company shows that over a period of one year the records indicate that it saved $1,244 in head injuries through the use of protective hats among 2,113 employees.

Another coal-mining company reports that among 250 underground employees and 43 surface employees they have realized a saving of $9,972 by decreasing head injuries and of $1,848 by decreasing foot injuries through the use of protective hats and safety shoes. A third coal-mining company reports that among 140 underground and 60 surface employees it saved in compensation and medical cost $2,967 in one year through the use of protective hats, shoes, and goggles. In some States the compensation cost for the loss of one eye is $2,000, an amount which would buy many goggles. These examples would indicate that any mining company can feel assured that money spent in furnishing its employees with protective clothing very probably will be more than repaid in decreased accidents, lower compensation rates, savings in compensation and medical care, and in a better functioning organization.

A recent announcement of the Pennsylvania Compensation Rating and Inspection Bureau indicates that a credit of 8 cents per $100 of pay roll will be allowed where all underground employees wear approved types of protective hats or caps. Under this schedule a credit of 5 cents is made for the use of goggles and a credit of 5 cents for the use of safety shoes, where all employees wear this type of protection as specified.

4. Value to miner. - The miner can expect fair value for money paid for protective clothing in addition to the protection it affords him. Protective hats will outwear most of the ordinary mining caps; protective shoes are no more expensive than ordinary work shoes, and in many instances where they are sold by the company direct to the workmen they are actually cheaper.

5. Voluntary or Compulsory use. - Whether the adoption of protective items of clothing should be voluntary by the workmen or made compulsory by the company depends upon the company organization and the type of workmen. It also depends to some extent upon the answer to the question "Who should pay for protective clothing?" Voluntary wearing of the equipment is highly desirable and can generally be obtained through education and example, both of which are entirely in the control of the operator. If the employee will not take steps to protect himself, however, then adoption of protective clothing should be made compulsory and strictly enforced.

SPECIFICATIONS

The Bureau of Mines has no specifications covering the design of hats, goggles, gloves, and shoes, nor is it likely to formulate schedules of tests or give approval to these items of wearing apparel. Some mining companies have given various makes of hats thorough trials to determine which is the most suitable for their conditions. Definite specifications for safety hats or caps, goggles, and shoes have been set up by the coal mine section, Pennsylvania Compensation Rating and Inspection Bureau, Harrisburg, Pa., as follows:

Specifications for Safety Hats or Caps

Definition:

The term "safety hat or cap" shall mean any hat or cap, by whatever name it may be known, that is made of such material and in such a manner so as to afford protection to the head from blows, falling objects, electric shock, etc., and be worn by men at work with a reasonable degree of comfort.

A - Types:

The Bureau believes the responsibility for selecting the particular type of hat or cap to be used should rest with the coal operator. The following types will be accepted, provided they meet the specifications set out below:

A-1. Hats or caps with a complete brim.
A-2. Hats or caps with one or more visors.
A-3. Any combination of a hat or cap.

B - Materials and Workmanship:

B-1. Materials used in the manufacture of a safety hat or cap shall be suitable for the particular type, light in weight, water-resisting, acid-resisting, and durable.

B-2. The materials of the lining and the hammock or cradle shall be of the best quality and strength. The assembly shall be so as to give the maximum protection and comfort to the wearer.

C - Construction:

C-1. All parts of a safety hat or cap shall be so assembled that those parts coming in contact with a man's head shall have a dielectric strength to withstand 750 volts for at least one (1) minute.

C-2. The lining of a safety hat or cap shall be so assembled that there shall be a cradle or hammock to support the complete hat or cap on the wearer's head. This cradle or hammock shall be constructed so as to be fully adjustable. When this cradle or hammock is properly adjusted, there shall be at least one (1) inch of space between the top of the wearer's head and the top of the inside of the crown. It shall be affixed to the hat or cap in a strong and substantial manner.

C-3. The safety hat or cap shall be so constructed as to comply with the following test: When mounted on a wood block similar to the position on a man's head, it shall withstand a pressure blow from an 8-pound iron ball (approximately 4 inches in diameter)

dropped vertically on to the center of the crown from a height of five (5) feet without denting or breaking sufficiently to touch the wood block. The cradle or hammock shall withstand the impact from this same blow without breaking or forcing hat or cap down over the head.

C-4. All safety hats or caps shall be fire-resisting.

C-5. The safety hat or cap shall be so designed and constructed as to give the required protection and be worn with a reasonable degree of comfort.

Specifications for Safety Goggles

Part I. Eyecups and Spectacle Goggles.

Part II. Wire Screen Goggles.

Part III. Use.

Part I. Eyecups and Spectacle Goggles

A - Types:

The Bureau believes the responsibility for selecting the particular type of goggle to be used should rest with the coal operator. The following types will be accepted, provided they meet the specifications set out below:

A-1. The cuptype goggle and/or its modifications.

A-2. The spectacle-type goggle and/or its modifications.

B - Materials and Workmanship:

B-1. Materials used in the manufacture of goggles shall be suitable for the purpose, of enduring quality, capable of being sterilized without deterioration, and non-irritating to the skin when subject to perspiration.

B-2. All metal materials used shall be corrosion-resistant.

B-3. Nitrocellulose or materials equally inflammable shall not be used.

B-4. Materials used shall be such as to combine mechanical strength and lightness of weight to a high degree and workmanship shall be the best throughout.

B-5. Where cup-type goggles are used, the cups shall be ventilated. The edge of the eyecup shall form a cushion at least 3/16 inch in facial contact width and shall be so designed as to prevent cutting of the face. If metal eyecups are used, the facial contact edges shall be bound with a cushion at least 3/16 inch in facial contact width and shall be so designed as to prevent cutting of the face by contact with the metal cup edge. This binding shall be non-flammable.

B-6. Head-bands shall be of good grade, at least 3/8 inch wide. They shall be adjusted to length and shall be fastened to the eyecups or goggle in such a manner as to be easily replaced and shall not interfere with the facial fit of the eyecups.

C - Lenses:

C-1. All lenses shall be made from a single solid glass plate of a quality suitable for optical use and shall be capable of passing the test as to quality, visibility, and shock resistance of the Bureau of Standards of the Federal Government.

C-2. The strength of all lenses shall be such as to comply with the drop-ball tests as required by the Bureau of Standards in the Federal Government specifications.

C-3. The optical surfaces of all lenses shall be free from visible surface defects. They shall be ground or polished on both sides. They shall not be effectively out of parallel. All lenses shall be suitable for continuous use by the wearer without discomfort or optical impairment to the normal eye.

C-4. All lenses in every type of goggle or eyecup shall bear some permanent distinctive marking by which the manufacturer may be readily identified.

D - General:

D-1. The specifications herein set out as Part I are abridged from the Federal Government's "Federal Specifications for Goggles: Eyecup, Chippers" as set out in Circular GGG-G-501, December 9, 1930. Any question of dispute that may arise from the interpretation of these standards shall be decided in accordance with the detailed specifications as set out in the Federal Government specifications referred to. It is recommended that persons interested in the use of goggles write to the Superintendent of Documents, Washington, D. C., and procure a copy of these Federal specifications. The price is $.05.

Part II. Wire-Screen Goggles

Note:- It has been found by experience that wire-screen goggles have not been a permanent installation in either mining or industries other than mining where goggles have been used for some long time. Usually, it is the first type considered. Probably the low initial cost may contribute to this. Many users of goggles who originally started with wire screen goggles later discarded the same for the spectacle or eyecups. However, the Bureau feels at this time that the wire-screen goggle should not be excluded, even though the other types are preferred, and therefore the following specifications covering wire-screen goggles are submitted:

A - Types:

A-1. Cup type.

A-2. Spectacle type.

A-3. All-wire type.

B - Materials:

B-1. The material used in wire-screen goggles shall be of noncorrosive wire not more than 0.016 inch in diameter and shall have not less than 20 strands to the inch.

C - Construction:

C-1. Cup-type or spectacle-type wire-screen goggles, other than the lens part, shall conform to the specifications approved for eyecups and spectacle goggles. The wire screen shall be inserted into the eyecup or spectacle frame, as the case may be, in such a manner that the ends of the wire cannot come in contact with the eye or face.

C-2. The all-wire screen goggle shall be die cut and shaped to fit the contour of the face. It shall have a firm binding around the edge so that there may be no danger of loose wire strands coming in contact with the eye or face. The head bands shall be adjustable.

Specifications for Safety Shoes

Definition. The term "Safety Shoe" shall mean any shoe, boot, or pac that has a protective toe cap inserted in the toe so as to give a reasonable amount of protection to the foot.

A - Types:

The Bureau believes the responsibility for selecting the particular type of safety shoe to be used should rest with the coal operator. The following types will be accepted, provided they meet the specifications set out below:

A-1. Leather shoe, boot, or pac.

A-2. Rubber shoe, boot, or pac.

A-3. Any combination of leather and rubber shoe, boot, or pac.

B - Materials and Workmanship:

B-1. Materials used in the manufacture of 'safety shoes' shall be suitable for the particular type and of high grade so as to give the maximum service.

B-2. The protective toe cap shall be constructed of material which will not be affected by heat or moisture conditions.

C - Construction:

C-1. The protective toe cap shall be so inserted in the shoe that it becomes an integral part of the shoe and shall not be a floating cap or what may be styled an appendix to the original shoe.

C-2. The protective toe cap shall be arched or shaped and inserted so as to withstand at least 350 foot-pounds pressure without crushing.

C-3. The protective toe cap shall be large enough to cover at least three (3) toes.

Note: Some protective toe caps are arc-shaped and long enough to cover all five toes.

REPORTS OF LIVES SAVED OR INJURIES PREVENTED

During the past few years the Bureau of Mines has received scores of reports indicating that lives have been saved or injury prevented through the use of protective clothing in mines. A few typical examples of these reports are given here because it would be impracticable to enumerate all of them in a report of this kind. It is impossible to estimate the number of injuries prevented by the use of protective clothing; unless an injury is received, the occurrence of a near-accident does not get into the record through the medium of an accident report and is, therefore, seldom known except to the person affected.

1. At - - - - - , the life of Mike - - - - - , machine runner, was saved because he wore a "hard-boiled" hat. Mike was struck on the head by a slab of coal weighing nearly 800 pounds and escaped with only a bruised leg and stiff neck. The impact of the falling slab was great enough to knock him about 8 feet from where he was working.

2. An underground pipeman, Joe - - - - - , at the - - - - - mine, was working around the starting apparatus of an electrically operated pump, and in leaning forward his lamp came in contact with an open circuit-breaker in such a manner as to "short" the circuit breaker and cause a flash that destroyed his lamp. . . . Because of the insulating qualities of the hat, Mr. - - - - suffered no injuries or inconvenience other than the loss of his lamp.

3. A serious head injury was prevented in one case when a large piece of ore dropped from the back and struck the miner on the head. The top of the hat was caved in, but the miner received only a slight scalp wound.

4. At the - - - - - mine, February 5, 1930, Matt - - - - - , a driller, was wearing a pair of - - - - - goggles while breaking chunks of hard ore. A piece of ore flew up and hit one of the lenses . . . . The lens was hit almost in dead center, but remained intact in the rim; he was uninjured.

5. Two serious foot injuries were prevented by the use of hard-toed shoes; in both cases a large chunk of ore rolled down from the top of a dirt pile and struck the miners on the feet.

6. A man was dumping cars at a station. In righting the car, the body came out of its proper place and caught the man's foot against the frame. Although this was a lost-time accident, it was felt that the severity was considerably lessened by the use of hard-toed shoes.

7. One wheel of a coal car ran over a miner's foot. The hard toe of his shoe was considerably dented, but the foot was uninjured.

8. A coal miner was completely covered by a fall of slate, and although he suffered a fracture of the leg and numerous bruises, his head was not injured, due to the protection of a safety hat. In this case, the safety hat prevented what might otherwise have been a fatal accident.

9. A car dropped off the track on to the foot of a snapper. One toe was crushed so badly that it had to be amputated. However, had the snapper not been wearing safety shoes, it would probably have been necessary to amputate all the toes or part of the foot.

RESULTS ACCOMPLISHED BY THE USE OF PROTECTIVE CLOTHING

Obviously, it is impossible to procure complete records showing the results accomplished by the use of protective clothing; however, the results accomplished by a few companies whose records are available, show conclusively that the use of such clothing pays big dividends.

A large eastern coal company operating three mines and employing about 1,600 men has made a remarkable reduction in eye and toe accidents during the past 4 years through the use of goggles and safety shoes. Safety hats have been introduced at these mines during the past year. The company did not enforce the use of goggles and safety shoes during 1929, but beginning in January 1930 every man entering the mines and all outside workmen were required to be equipped with safety shoes and goggles. The following tabulation shows the results that have been accomplished by this company in 3 years' time:

Accident reduction effected by wearing goggles and safety shoes

Eye Accidents

Year	Accidents	Days lost time	Frequency rate	Severity rate
1929	396	5,839	60.40	0.889
1930	150	3,793	24.42	.617
1931	55	79	14.43	.021
1932	17	13	5.56	.005

Toe Accidents

1929	164	1,129	25.02	.171
1930	72	510	11.72	.082
1931	20	530	5.27	.139
1932	5	54	1.93	.021

All accidents, including those in which no time was lost, are included in the foregoing tabulation. It is noted that the frequency of eye accidents has been reduced 91 percent and the severity has been reduced 99 percent. During these 4 years the record shows a reduction in the frequency of toe accidents of 92 percent, with a reduction in severity of 93 percent.

An anthracite company introduced safety hats at its mines; during 1929 about 10 percent of the employees were so equipped, but by 1932 about 95 percent were provided with safety hats. The use of safety hats at this property has reduced head injuries as follows:

Accident reduction effected by wearing hard hats

Year	Head injuries	Days lost
1929	126	693
1930	142	1,519
1931	28	123
1932 to June 30	7	26

At an Ohio mine the accident record showed 16 lost-time accidents due to toe injuries during 1930. A safety campaign to induce all employees to wear safety shoes was started, and during 1931 only six toe injuries occurred; in 1932 when all the men were equipped with safety shoes, only one toe injury occurred.

A large anthracite company employing more than 11,000 men had 775 compensable eye injuries during 1931--12 percent of all the injuries for that year. During the latter part of the year, an educational campaign was started to teach the men the necessity of eye protection, with the result that 91 percent were provided with goggles and eye accidents were reduced more than 50 percent during 1932.

This company has reduced head injuries 75 percent within 2 years by the use of safety hats and has reduced toe injuries more than 50 percent during the same period by the use of safety shoes.

In a group of seven coal mines whose employees are nearly all equipped with safety hats, safety shoes, and goggles, some remarkable reductions have been made in the frequency and severity of head, foot, and eye injuries. An average reduction of 75 percent in the frequency of head and scalp injuries, a reduction of 96 percent in the severity of these injuries, has been made since the introduction of safety hats. The frequency of foot and toe injuries has also been reduced 75 percent, while the severity of these injuries has been reduced 93 percent. The frequency of eye injuries has been reduced 68 percent, while the severity has been reduced 43 percent. By the use of protective hats, shoes, and goggles, three of the companies of this group have practically eliminated all head, foot, and eye accidents in recent months.

The experience of copper and iron mining companies in the Lake Superior district has led to the more or less universal adoption of protective clothing as a safety standard; in many of the mines, the rule as to safety hats, shoes, and goggles is extended to visitors, no one being allowed to go underground without this equipment.

The combined records of six large mining companies in the Lake Superior district whose employees are all equipped with safety hats, shoes, and goggles show that the use of such equipment is worth while. An average reduction of 68 percent in the frequency of head injuries and a reduction of 89 percent in severity of such injuries have been made since the introduction of safety hats. The frequency of foot injuries has been reduced 65 percent, with a reduction in severity of such injuries of 53 percent. The reduction in eye injuries in this district has not been as great, on the average, as it has been in some other districts, the frequency of eye injuries being reduced 6 percent and the severity 33 percent.

CONCLUSIONS

Both coal and metal mining companies are realizing the value of protective equipment, the wearing of which by all of their employees is required by many companies. With the increased use of protective clothing, a general reduction in the accident records may be expected.

The miner, the company, and the general public benefit by the use of protective clothing - the miner by avoiding the suffering from an injury and the loss of wages, the company by savings made in compensation and medical cost and by increased efficiency and morale, and the public by the decrease in costs of various kinds consequent to preventing men from being killed or maimed.

Statistics show that about 15 percent of the compensation cost in the bituminous mines of Pennsylvania during a 5-year period could have been prevented had the victims worn protective clothing. These statistics will probably apply, on the average, to mines in other States. Thus it is apparent that the saving in compensation alone is well worth the investment in protective clothing.

Although this report refers specifically to coal and metal mines, what has been said applies equally to nonmetal mines and quarries and also to certain "jobs" in mills and smelters. Many accidents in open-pit operations would have been avoided or lessened if goggles, hard shoes, and gloves had been worn. Head protection of some sort is also necessary in many if not most such types of work.

I. C. 6725

June, 1933

Department of Commerce

United States Bureau of Mines
Scott Turner, Director

INFORMATION CIRCULAR

EXPLOSIVES ACCIDENTS IN CALIFORNIA METAL MINES

BY

S. H. Ash

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DEPARTMENT OF COMMERCE --- BUREAU OF MINES

EXPLOSIVES ACCIDENTS IN CALIFORNIA METAL MINES[1]

By S. H. Ash[2]

On December 2, 1932, the Governor of California called upon the representatives of the mining industry of the State to attend a meeting at Sacramento to discuss ways and means of assisting in relieving some of its perplexing and extremely distressing problems, particularly the mounting cost of accidents which faces the industry with an industrial insurance rate for classification of $11.85 per $100 of pay roll, effective January 1, 1933. The significance of this fact is at once apparent when it is realized that the classification rates per $100 of pay roll for mining have steadily increased for some time, as follows: $5.81 in 1924, $6.54 in 1925 and 1926, $8.04 in 1927, $8.53 in 1928, $9.05 in 1929, $9.42 in 1930, $10.54 in 1931, $10.99 in 1932, and $11.85 in 1933. In other words, the cost rates have more than doubled, while statistics indicate that the accident rates have shown an entirely different trend.

At the meeting mentioned, one of the management officials sounded an effective keynote when he declared that a spade should be called a spade. The perplexing problems of any industry can be approached best and remedies applied if that policy is kept to the front. It is appropriate to mention that high accident rates and a decrease in wages in recent years have unquestionably accounted in part for a high unit cost rate on a pay-roll basis; it also is well to explain that the policy years[3] 1926 to 1930, inclusive, are the years on which the accident-cost experience for 1933 is based. That accident rates have not undergone the expected beneficial change in California's experience is apparent in the words of a Bureau of Mines report:[4]

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1 The Bureau of Mines will welcome reprinting of this article, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6725."

2 District engineer, safety division, U. S. Bureau of Mines, Berkeley, Calif.

3 A policy year includes all of the policies written during the designated year but refers to experience developed in the following calendar year.

4 U. S. Bureau of Mines, Accident-Prevention Record of the Mining Industry in California in 1931: Health and Safety Survey, Demographical Division, Oct. 26, 1932.

The safety record for mines in California in 1931 was better than that for 1930 and for most years during the past decade, according to figures just compiled by the United States Bureau of Mines from reports received from operating companies through the Industrial Accident Commission of California. Records for the State indicated an average rate of 109 accidents per million man-hours of employment as compared with 120 in the previous year. On only three occasions since 1921 has the accident rate for California mines been more favorable than it was last year, these instances being in 1927, 1928, and 1929.

The figures prepared by the Bureau of Mines for 1931, which covered all important mines and numerous small operations, showed a total of 640 mines employing 5,553 men who worked a total of 10,755,493 man-hours; this was 10 percent below the number of hours worked in 1930. Each employee worked an average of 241 days or 1,940 hours. The average working time in the preceding year was 251 days or 2,014 hours per employee. Accidents caused 15 deaths and 1,160 nonfatal injuries.

According to the Industrial Accident Commission's classification, 19 men were killed during 1931. Of these deaths, three, or 15.78 percent, were caused by explosives--a slightly higher rate for explosives than in 1930.

The following list (table 1) of fatal accidents in the California mining industry in 1932 were reported to the Industrial Accident Commission:

Table 1.- Fatal accidents in mining in California during 1932

Date of accident	Kind of mine	Number employed	Number killed	Cause of accident
Jan. 28	Gold mine.	Over 50	2	Explosives (fuse and detonator blasting). Picked into missed shot in winze.
Feb. 13	do.	Over 100	3	Explosives (fuse and detonator blasting). Stayed too long. Wet fuse. In drift.
Mar. 23	do.	Less than 10	1	Caught in machinery on surface.
Mar. 30	Gold dredge.	Over 20	1	Fell from dredge; drowned; on surface.
Apr. 12	Gold mine.	Less than 10	1	Fell down shaft.
Apr. 12	do.	do. 20	1	Electrocuted. Signal wire charged (in winze) with 440 volts.
May 16	do.	Over 100	1	Struck by falling timber. Underground.
May 27	do.	do.	1	Fall of rock. Underground.
July 6	Gold dredge.	Over 20	1	Electrocuted on dredge.
July 30	Gold mine.	Less than 10	1	Explosives (fuse and detonator blasting) 7 holes. Carbide lamp extinguished. Stayed too long in winze.
Aug. 24	do.	3	1	Explosives (fuse and detonators) in drift. Primer exploded, container fell, transporting same.
Sept. 14	do.	3	1	Explosives (fuse and detonators) in drift. Stayed too long. Wet fuse. 2 holes.
Sept. 17	do.	Over 100	1	Explosives (fuse and detonators). Starting raise. Stayed too long. 15 holes.
Nov. 3	do.	do.	1	Skip and shaft timber.
Nov. 19	do.	Less than 5	3	Suffocated, lack of ventilation. Shaft.
Dec. 5	do.	Less than 10	1	Fall of roof in drift.
Dec. 5	do.	Over 50	1	Skip and shaft timber. Underground.
Dec. 28	do.	Over 10	2	Killed by cave caused by blasting (fuse and detonators used). Underground.
		Total	24	

Of the foregoing accidents it will be observed that six resulting in nine fatalities are directly assigned to explosives and that two fatalities are indirectly attributable to them; the fatal accidents assigned directly to explosives in 1932 caused 37.5 percent of the total fatalities in the metal mines of the State for that year.

It will be further observed that all blasting was done by fuse and detonators, and that two accidents with three fatalities occurred in winzes; three accidents which killed five men and injured three occurred in drifts; and one accident killed one man while he was starting a raise. According to table 1, explosives accidents lead the list of fatalities as to cause in California for the year 1932. And the majority of these accidents, if not all of them, could have been prevented by electric blasting.

A study of the occurrence of mine accidents and their cost in California indicates that there is a clear and vital distinction between the frequency and severity of accidents, and the cost of certain types of injuries; to these types of injuries belong those resulting from accidents caused by blasting or the use of explosives.

There will be no appreciable change in accident rates and the effect that they have on the mounting costs of accidents unless there is a drastic reduction in certain classes of accidents of comparatively low frequency but of marked severity; among these accidents belong those resulting from the use of explosives. Any reduction in the frequency and severity of accidents from explosives can best be brought about by a recognition of unsafe practices and a substitution of safer ones in their places.

In all types of mining, including coal mining, the second important cause of fatalities from mine accidents usually is haulage; in California, however, explosives as a cause of accidents takes second place. It is evident that explosives practice in the mines of California is not conducted in line with the safety standards set by mineral industries of the country as a whole. This is not due to the use of an excessive or unusual amount of explosives. Even in quarries where explosives are important agencies the accidents caused by their use take third place.

In the metal mines[5] of the United States during the 5-year period 1926-1930, the use, or possibly the misuse, of explosives caused 10.97 percent of the fatalities and 1.25 percent of the nonfatal accidents; in the metal mines of California during the same period 12.60 percent of the fatalities and 1.12 percent of the nonfatal accidents were caused by explosives.

Because explosives accidents in California are principally in operations directly or indirectly connected with the breaking of the ground, it is here that changes must be made in current practice if relief is to be obtained. For this reason this report deals primarily with accidents connected with the loading and firing of holes. Fatal accident rates are more dependable as to accuracy than are nonfatal accident rates, particularly where small mines are included; many nonfatal accidents are not reported, but fatalities are brought

5 Adams, William W., Metal-Mine Accidents in the United States: Bull. 362, Bureau of Mines, 1930, 99 pp.

to attention through various means, including the newspapers in cases of small mine accidents.

SEVERITY AND COST OF EXPLOSIVES ACCIDENTS

One of the outstanding facts connected with explosives accidents is their relatively great severity and resultant high accident cost. Although for the period 1926-1930 explosives accidents accounted for only 1.12 percent of the nonfatal accidents, taking eleventh place as a cause of such accidents, they account for a high proportion of the permanent total and permanent partial disability accidents. An explosives accident is likely to result in fatalities or serious injuries or both, and the number of blasting accidents at some mines indicates lax blasting practice in those properties.

The importance of reducing explosives accidents is particularly emphasized in a study of mine-accident costs. Although this type of accident accounted for nearly 17 percent of the fatalities over a considerable period, the accident cost was only 2.1 percent of the total, on account of the fact that single men predominated among the number killed and compensation is much less for a fatality to a single man than to a married man. The severity of the nonfatal accidents from explosives and blasting is striking; eighteen nonfatal accidents, which are only 1.07 percent of the total, cost $115,205 (compensation and medical), or 19.4 percent of the total cost of all accidents. The cash outlay for industrial insurance alone, however, on account of the insurance cost in these instances, was approximately $194,000.

Permanent partial disability injuries constitute 3.87 percent of the total number of all mining accidents in California and 28.75 percent of the total cost of all such accidents. The same class of injuries from explosives accounts for only 0.42 percent of the total number of mining accidents, but accounts for 12.68 percent of the total cost of all such injuries. This type of injury is of such nature that no malingering can be done. Explosives thus appear to be responsible for approximately 44 percent of the cost of permanent partial disability injuries in mining in California. Electric blasting has played no part in these injuries, so far as the writer is informed, as very little if any blasting has been done by electrical methods. However, "staying too long," "short fuse," "premature blasts," "returned too soon," and "delayed blasts," with "missed shots" form the familiar list of causes, involving the use or possibly the misuse of blasting with fuse and detonators. Out of 169 permanent partial disability accidents in California's mining industry in the period 1927-1930, eye injuries are of frequent occurrence, as they amounted to 41, or nearly 25 percent of the accidents of this class. Many of these injuries resulted from explosives or from flying particles raised by explosives. Of the 41 injuries, 22 were sustained by miners, 4 by muckers, 3 each by laborers and jackhammer men, 2 each by shift bosses and dredgemen, and 1 each by a shaft man, blacksmith, loader, boiler repairman, and carpenter.

SIZE OF MINE AND PERIOD OF OPERATION

In any accident discussion of California mining experience the size of the mine is a matter of importance. In 1930 California had 476 active operations at which 5,941 men were employed, indicating an average of 12.5 men per operation. A detailed study shows that a mine employing 10 men is a large one in California. In the United States as a whole, statistics indicate that in 1930 only 7.32 percent of the total number of underground man-days were performed at mines employing 1 to 19 men, which, however, constituted 79.2 percent of the operations, whereas in California 22.25 percent of the total number of man-days were performed at mines employing 1 to 19 men, mines of this size constituting 84.3 percent of the operations. Statistics[6] also indicate that for any given class of mines graded according to number of men employed, those which operate continuously generally have better safety records than those that do not, and that larger mines usually have fewer accidents than smaller ones in proportion to the number of men employed. Coordination of effort, an appreciation of a mutual problem, and widespread education, especially insofar as blasting practices are concerned, are essential if accidents in California mines are to be reduced. As part of a movement toward accident reduction, the industry during the past year inaugurated a campaign of first-aid instruction which has resulted in complete or 100 percent training at some 26 mining operations, a total of 1,585 men being trained by the United States Bureau of Mines.

The effect of explosives accidents is most emphatically shown in study of mine accidents in a district involving a large number of small operations. The revival of gold mining in 1929, 1930, 1931, and 1932 has reopened a number of small properties, where in general the explosives practice has been anything but safe or efficient. Investigations of explosives accidents at some comparatively small operations and also at some large ones show that blasting at small operations almost invariably involves the use of fuse and detonators, and with the lack of inspection, supervision, and care usually found in small mining operations, blasting accidents become numerous and are likely to be severe in their effects.

In California during 1930, a total of 390 metal and nonmetallic mines employed 6,243 300-day workers who worked 1,873,148 man-days. Gold mining accounted for 72.57 percent of the operations and 56.08 percent of the man-days; the respective figures for nonmetal mining were 14.86 percent and 21.23 percent, for quicksilver mines 8.73 percent and 7.52 percent, and for copper mines 2.05 percent and 14.55 percent.

There are relatively few large mines in California in comparison with some States. A group tabulation for 1930 shows in general that the average loss of time per accident is lower at the large mines than at the small mines, at which the severity rate particularly is much higher. However, the smaller mines appear to have much fewer nonfatal accidents per 1,000 shifts worked than do the larger mines; hence, the accident-frequency rate of the small mines seems better than that of the larger ones, at least for the year 1930.

6 Adams, William W., Work cited.

Because of the closer regulation of blasting in large mines, individual blasting accidents generally occur more often at small mines, particularly those that have been or are being reopened after a shut-down of several months or years. Of 80 fatalities in a group of small mines in California during the years 1894 to 1909, inclusive, 24 were caused by explosives, 20 by falls of rock, 17 by falls of persons, 13 by hoisting, and the other 6 by miscellaneous causes. Of the 24 listed as caused by explosives, loading and firing holes killed 10, missed holes 9, transportation 3, and striking explosives in the muck pile with a hammer, 2. This seems to indicate that the mistakes of 25 years or more in the past are largely of the same type as those which occur today; evidently little progress has been made, at least in some phases of blasting in the metal mines of California.

During the period 1925-1930, according to the Industrial Accident Commission of California, miners suffered 171 fatalities, of which 27 were caused by explosives: explosion, 15; premature blast, 5; overcome by fumes, 3; drilled into missed hole, 2; and 1 each for delayed blast and picking into missed hole.

The following descriptions cover some recent explosives accidents in California's metal mines:

1. On February 13, 1932, 35 shots of a round of 47 shots in a drift of a large California gold mine had been lighted and 12 had not been lighted, but because four men remained too long at the place, three of them were killed and one was seriously injured. Two of them were shovelers who were standing by with carbide lamps for emergency. There had been trouble in spitting the fuses because they had become wet.

Holes were drilled 5 to 6 feet deep. Forty percent gelatin dynamite was fired by No. 6 or 8 detonators, the latter in this case. The fuse was triple-taped, water-safety fuse used for wet ground and for holes under water. Detonators were crimped to the fuse on the surface, 7-foot lengths of fuse being used for drifts. At shooting time it is customary for two men to light the fuses and two to stand by with carbide lamps. Having two men to stand by with lights is usually considered ample precaution to protect shot lighters using carbide lights, but in this case the precaution merely added additional casualties. The evidence showed that the fuse became damp and it was necessary to cut several of them, which caused delay. Electric blasting was adopted following this accident.

2. A similar accident occurred at another large mine in this State on September 17, 1932, whereby one man working in a raise lost his life trying alone to light 22 holes.

3. On April 14, 1932, while two men in a winze in a small California gold mine were loading waste an explosion took place which for some unexplained reason did them no injury. The winze is 5 by 10 feet in section and 24 holes had been fired of which three were reported as misfires. Forty percent gelatin dynamite with triple-tape fuse and No. 6 detonators had been used.

Following this incident the shots were fired electrically for a time. Shots were connected in series, using delay detonators, and fired by a 50-shot blasting machine; but because of misfires, power blasting in parallel was being considered. On July 30, 1932, two men were sinking a winze and instead of using the electrical method provided, blasted by fuse and detonators. The fuses of two shots were lighted by carbide lights, one of which was extinguished, and after five more shots were lighted, one of the men left. The shots went off and killed one man--a direct result of violation of safety orders and of common sense as well. Such accidents as this are likely to be unknown with electric blasting and are very likely to continue as long as fuse is used.

4. An explosives accident occurred at a small California gold mine on July 24, 1932, resulting in the death of one man and the injury of another. Three holes had been drilled in the face of the drift and three primers had been prepared in three half-sticks of dynamite on the surface; these primers and three sticks of dynamite were placed in a pail which was being carried into the mine at the time of the accident. What caused the dynamite to explode is not known, as there were no witnesses to the accident. The victim's legs were practically blown off below the knees and he was otherwise injured, indicating that possibly the pail of explosives was dropped, the impact causing the dynamite to explode. A box of explosives was stored on the surface and at the time was exposed to the intense heat of the sun; with this type of carelessness in the storing of explosives there is a possibility that undue exposure contributed to the sensitiveness of the explosives.

This accident, as well as others of a more or less similar nature, emphasizes the fact that the method of storage of explosives at some small mines is not conducive to safety, and it is also a good demonstration of the fact that the transportation of primers and explosives together is dangerous practice, although many mining people do not hesitate to carry detonators and explosives together and to minimize or even to deride recommendations against the manifestly dangerous practice.

5. An explosives accident occurred during 1932 at a California gold mine. A man had a contract for sinking a winze at a point about 170 feet from the shaft; four men were employed on the contract or lease, two on each of two shifts. One shift had blasted a round of holes using fuse and detonators; the other shift upon coming to work were informed that six holes had been reported as having missed, but the two men proceeded into the mine, mucked out the rock, and found five missed holes but did not find the sixth.

The superintendent came to the hoist house and upon finding that there were six missed holes, immediately went into the mine and was at the winze at 10 a.m. The contractor asked if he could reblast the missed holes immediately, and the superintendent ordered him not to blast before noon as the smoke would interfere with the workers in an adjacent stope. The superintendent left the winze about 10:30 a.m. and had just reached the shaft station, about 170 feet away, when a report was heard. He and the shift boss ran back and assisted in removing the two men, one of whom was dead.

The holes were in wet ground. Although what is classed as good fuse was being used in this place, one or more misfires were occasionally reported. Following this accident it was decided that all blasting in this winze was to be done electrically; electric exploders had been used formerly by other operators, but their use had been discontinued.

6. On September 14, 1932, a fatal accident caused by explosives occurred at a small California mine. By this accident two men were injured, one fatally. A third man stationed at the portal was not injured.

The writer went to investigate this accident, and was directed to the mine by two small boys. This fact is mentioned for the reason that on arriving at the mine a roll of waterproof, triple-tape fuse and some No. 6 blasting caps were in full view of the portal. There was no one around. It is remarkable that more children are not killed and injured at places where their inquisitiveness is subjected to such temptation. The writer has visited several small mines at which explosives and explosives appliances are allowed to lie around in a way that exhibits extreme carelessness and lack of regulatory measures to prevent serious accidents from indiscriminate use of such materials.

This small mine is a typical one opened in gold-bearing gravel. About 15 feet from the edge of the brush a few three-piece sets hold the ground open. The adit is then driven about 3-1/2 to 4 feet wide and 4-1/2 to 6 feet high for a distance of about 350 feet and is crooked and untimbered. The last 50 feet of the adit was so wet that the flame on a candle could be maintained only with difficulty and fuse would be difficult to light. Two fuses, approximately 3 feet long, found at the portal show that trouble had been previously experienced; the fuse had burned to the detonators but had failed to explode.

In this instance it was plain to see what had happened. Two short holes 2-1/2 to 3 feet long were drilled in the soft, wet gravel and sand formation at the face, which was about 16 feet in cross section; one hole on the left side was standing, but evidence showed that the one on the right had gone off and caused the damage. The two holes were each loaded with one stick of dynamite (strength could not be ascertained) as a primer, using No. 6 detonators and 3 feet of so-called waterproof fuse. It was learned that the injured man was standing about 10 to 15 feet behind the man who was killed. The latter, after lighting the first hole with his carbide light, had much difficulty in trying to light the second because the fuse became wet; as the fuses were only 3 feet long, he had very little time and as has occurred in numerous similar cases he remained too long. This accident undoubtedly would have been prevented had electric blasting been in use.

The mine safety orders of California contain many provisions which, if followed, would eliminate most explosives accidents. A study of accidents in which more than one man has been killed indicates that the safety orders are being disobeyed in many instances.

At some of the large mines the explosives storage facilities, both surface and underground, are in accordance with the best general practice, whereas at some others there is not even a pretense of safe or sane practice at times. Unquestionably, poor practice in storage of explosives is responsible for many of the accidents in connection with the use of explosives, and especially is poor practice responsible for misfires, premature explosions, and gassing from blasting fumes.

Accidents from transportation of explosives have not been of relative importance, but precautions against them can be much improved, especially at small mines.

Probably no greater opportunity to reduce the more serious types of accidents in California mines is offered than by adopting more approved blasting methods. With the exception of a few instances in sinking shafts and winzes and in driving tunnels, blasting is usually done with fuse and detonators, and by the miners themselves, with some assistance at times from the shift boss.

Experience in California with blasting by fuse and detonators demonstrates that 45 minutes as provided by law permits unsafe interpretations and abuses, and is too short a period before returning to a missed shot. A provision for an 8-hour interval where missed shots are suspected when using fuse and detonators would, if enforced, aid materially in bringing about safer and more efficient blasting. This interval is provided by law in some States, and some States prohibit return to a missed hole until the next day. The interval between blasting and returning to a shot that has failed to fire should be at least 2 or 3 hours under any system involving the use of fuse and detonators, and in fact with any kind of blasting in which the present-day types of explosives are employed.

Of the total number of fatal accidents in California mining for the period 1925-1930, 1.76 percent resulted from suffocation by powder smoke; a major disaster with a loss of 7 lives occurred from this cause in 1917. These accidents are a result of returning too soon after blasting, use of improper explosives which produce gases or of explosives which have been stored improperly, blasting of holes which have not been tamped with inert material, or possibly of inadequate ventilation. Although electric blasting will not entirely prevent this type of accident, it will help materially to eliminate fumes from blasting[7] by reducing the poisonous gases from burning fuse and burning explosive, in addition to eliminating other unsafe practices.

Improper charging and overloading not only cause dangerous quantities of poisonous gases and premature shots but also cause delayed shots.

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7 Harrington, D., Data on Metal-Mine Ventilation in 1929: Inf. Circ. 6246, Bureau of Mines, February 1930, p. 7.

In open-pit work in California where the holes are sprung with dynamite and then blasted with black powder, accidents have occurred from charging the hole too soon after springing when the heat in the bottom of the hole was sufficient to ignite the black powder.

No type of mine accident can be more easily reduced than can blasting accidents by substituting good blasting practice for unsafe practices. If good practice is backed up by intelligent instruction at the face, by effective supervision, and by rigid but just discipline for violations, there is no doubt that considerable suffering can be eliminated and economic advantages gained in California mines.

AN EXAMPLE OF CHANGE IN EXPLOSIVES PRACTICE

Because sections of California[8] have been found to be gassy, including that section in which the tunnels[9] of the Hetch Hetchy water-supply project of the city and county of San Francisco are being driven, blasting practice on that project represents as nearly a maximum degree of safety achievement as is obtainable under existing circumstances.

On account of the nature of the work involved in tunnel driving the use of explosives is one of the major operating items. Unsafe blasting practices offer the greatest hazards in the average tunnel for large groups of the tunnel workmen. Blasting with long rounds in most ground and with short rounds in seamy and blocky ground is common tunneling and ore-mining practice. The use of explosives on the Hetch Hetchy project represents a transition from the former unsafe crowding practices of contracting work when using fuse and detonators, to the more modern and much safer method of using permissible explosives and electric blasting. Probably no better example of what can be accomplished in safe blasting practice is to be found than in the experience of the Hetch Hetchy project.

Between the years 1920 and 1923, four explosions caused by explosives and in which the use of fuse and detonators was involved cost the lives of 11 persons and inflicted injuries on 10 others; these explosions occurred during the driving of approximately 85,000 feet of the Hetch Hetchy tunnel. Following this period, electric blasting was adopted, since when there has been only one explosive accident, involving a loss of one life, caused by tamping an electric primer; an inert primer would have prevented this accident. During the period 1924 to February 1, 1933, 227,080 feet of tunnel has been driven.

There were eight surface magazines at the respective sections on the Coast Range division of the tunnel. These are built to conform to California laws of distances and are of bullet proof construction.

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8 Ash, S. H., and Rankin, J. H., Permissible Electric Cap Lamps and Ventilation in Certain California Mines and Water-Tunnel Construction: Bull. 359, Bureau of Mines, 1932, pp. 2-4.

9 Harrington, D., Progress of Metal-Mine Ventilation in 1930: Inf. Circ. 6469, Bureau of Mines, 1931, pp. 5-12.

The explosives are taken into the tunnels by the motorman. Two-compartment, insulated powder cars (permitted by law, but unquestionably not the safest practice) are provided for the explosives and detonators, which are kept separate. The transportation of primers is always attendant with some danger.

Not more than 100 pounds of explosive is stored underground. It is kept in locked wooden boxes. This explosive is sufficient for single-shot plugging only, which is done with permissible explosive and permissible single-shot blasting units. Explosive sufficient for the entire round up to 40 holes is brought into a tunnel from the surface.

Explosives of the Gelatin-Colites and Herco-Gels have been used since 1930; electric detonators of No. 8 strength and delay types are used. Explosives and detonators are not handled by the same man at the same time. The miners and shift boss load the holes. Cartridges are used and care is taken to guard against air-spacing. The cartridges are slit and the explosive is carefully rammed. Clay stemming is used and a wooden-type bar is used in tamping. The primer is placed as the second stick in the bottom of the hole.

Rounds of 20 to 40 holes are drilled with compressed-air drills using water. Four to seven sticks of 1-1/8- by 8-inch permissible explosive are used per hole and from 5 to 9 feet are "pulled," depending on the ground.

Shooting is done by a fire boss who has satisfied the Industrial Accident Commission as to his eligibility to detect gas and to take precautions against gas ignitions; one fire boss is employed on each shift. One to 40 shots are fired at one time, with delay-action exploders.

Power blasting in parallel (440 volts a.c.) is used for the main rounds, a special blasting switch[10] controlling a momentary circuit of very short duration being used for this purpose. The locked switch is operated by the fire boss from the shaft crosscuts after all men have retreated about 5,000 feet or more from the face.

Misfires were common prior to the adoption of electric blasting, and are assigned as the cause of all previous explosives accidents, costing 11 lives. No fatalities have been caused by misfires since adopting electric blasting. Two nonfatal accidents occurred when old unexploded primers used for plugging, and carelessly not reported, were struck in picking for posts. Inert primers,[11] it is believed, would prevent this type of accident; and single-shot blasting for plug shots and parallel blasting with a power circuit have eliminated misfires.

An examination is carefully made after blasting to try to detect any misfires which may have occurred. In case of known or suspected misfires which are examined and reported, a new primer is used and the hole fired after the

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10 See footnote 8.

11 Manning, R. I. C., and Soule, Thomas, Electrical Blasting at the Morenci Mines of the Phelps Dodge Corporation, Morenci, Arizona; The Explosives Eng., vol. 9, June and July, 1931, pp. 209-212, 250-254.

stemming has been removed by water jet. The legs of all electric detonators are short-circuited and the short is removed following a check on the main line. It is impossible to close the circuit unknowingly, and opening the line switches shorts the blasting line at the outby point. No record of misfires is available; however, they are rare and appear to be a result of series blasting for plug shots when more than one shot was fired from a high-voltage power circuit.

The number of explosives accidents not involving a loss of life in the tunneling operations of the entire State is not known. The use of the inert primer in tunneling operations would aid materially in preventing explosives accidents from missed shots, loading holes, striking primers, removing charges, and transportation, and would be a safeguard in various hazardous conditions in tunneling and metal-mine operations. Such a primer appears to offer a feasible remedy for certain types of explosives accidents, whether they involve the use of fuse and detonators or of electric exploders.

SUMMARY OF EXPLOSIVES ACCIDENTS IN CALIFORNIA INDUSTRIES

The use of explosives has cost many lives and resulted in serious injuries in the mining, quarrying, and construction industries in California. A study of the explosives accidents in construction work indicates that for the same number of shots fired, accidents occur most frequently in tunneling operations, especially when fuse and detonators are used. The Industrial Accident Commission has a record of 133 explosives accidents resulting in a loss of life and serious injuries, 94 persons being killed and 46 injured. Table 2 shows some of the various factors involved in the 133 accidents, but does not include all accidents whereby no one was killed and only injuries resulted, or of other accidents not directly involved in direct blasting operations. In the California mining industry, for the period 1924-1930, there were 30 fatalities and 129 injuries (potential fatalities) from explosives from all causes. The blasting system in these instances was one involving the use of fuse and detonators. Until recently there has been no appreciable amount of electric blasting in Californian mines, although now it has been generally adopted in the larger and more important tunneling operations; but fuse and detonators continue to be used in smaller tunneling jobs.

Whether accidents would probably have been prevented if electric blasting methods had been used instead of fuse and detonators is debatable to some extent, but there are accidents classed as caused by premature explosions; missed shots; short, fast, wet, and delayed fuse; crimping caps; spitting too many; staying too long; returned too soon; counted too many shots; besides others, that unquestionably would not have happened if electric blasting had been employed.

Certain types of explosives accidents cannot be prevented under certain practices (in long rounds, shafts, winzes, long raises) unless electric blasting is used. The data given in table 2 represent the combined opinion of the Safety Department of the State Industrial Accident Commission and that of the writer.

Table 2. - Summary of typical explosives accidents in Californian industries

Accidents		133
Persons killed	94	
Persons injured	46	
Total killed and injured		140
Accidents:		
At mines	59	
At quarries	10	
On construction (tunnels principally)	51	
Miscellaneous	13	133
Accidents using fuse		130
Accidents using electric blasting		1/ 3
Injured and killed according to cause:		
Premature blasts	11	
Stayed too long	27	
Drilled or picked into missed hole	57	
Other causes (tamped too hard, short fuse, detonators, returned too soon)	33	
Cause unknown	12	140
Accidents that probably would have been less in frequency if electric blasting had been used		114

1/ Two at tunneling operations. One caused by tamping primer and the other by stray current.

GENERAL DISCUSSION

Mine Safety Orders 1743 to 1745, inclusive, pertain to explosives practice in California mines. The noncompliance, whether intentional or unintentional, with certain sections of these orders is in evidence almost every time an accident occurs. Obviously, repeated and uneventful violation of safety rules is conducive to a tolerant attitude toward unsafe practice; however, unsafe practice will sooner or later result disastrously. Three pertinent sections are part of Safety Order 1745, namely:

> The number of explosions in every blast, except in cases of simultaneous firing, shall be counted by the man firing the same, and if the total number of explosions is less than the number of charges fired, a report of the discrepancy shall be made as the

superintendent shall direct. When a blast has been fired and it is not certain that all the charges have exploded, no person shall enter the place where such charges were placed within forty-five minutes after the explosion.

(g) No man shall "spit" more than fifteen fuses at one time and should it be necessary to blast a greater number of holes than fifteen, he must have assistance.

(k) Electric exploders only shall be used in shaft sinkings, except when such use is not feasible.

Every one of these orders repeatedly has been found to be deplorably deficient. Experience would appear to indicate that they are based on average results rather than minimum requirements, and the wording of an order sometimes permits of a construction that inevitably leads to accidents. Fifteen fuses appear to be entirely too many for any one man to ignite, when alone, except under most favorable conditions. Experiences cited in this report confirm the fact that under some conditions even two shots are too many when using fuse and detonator. The exception in section (k) just quoted nullifies this section to the extent that it is almost universally evaded, as this report shows in part by examples of explosives accidents. If there is one type of accident the avoidance of which calls for a close interpretation and compliance with rigid orders, it is that relating to explosives practice, and the industry eventually pays the bills for errors of practice in human wreckage and economic distress.

Following the Mitchell shaft disaster of 1930 at the Hetch Hetchy water-supply project where 12 men were killed—coming as it did almost immediately after the Alameda Creek tunnel disaster (7 lives lost), and the Glenn mine fire when 5 lives were lost—a committee[12] was appointed by the former Governor of California. This committee, among other things, recommended that in California mines—

p. Where at all feasible, electrical blasting should supersede the prevailing use of fuse and detached "caps" or detonators. However, electrical blasting is by no means foolproof and there are definite hazards with delay action detonators, particularly where explosive gas may be found. Electric detonators should have the leg or leading wires short circuited on the surface before being sent underground, and this "short" should be maintained until practically ready to shoot. All blasting in sinking shafts, dip slopes, and winzes should be by electricity.

q. Explosives or detonators should not be stored at any point where an explosion would do damage to the exit for men, nor should they be left temporarily within 200 feet of any shaft station.

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12 See footnote 8.

If there are to be any changes in current practice, the remedy is going to come in large part, if not completely, from the mining industry itself. Probably no activity in mine safety practice has a wider distribution of informative literature than that pertaining to explosives and their use; every carton, every box, and every utility is accompanied with its usual admonishments and list of "don'ts." The solution of the problem of accidents from explosives is to adopt what is known to be safe and efficient practice.

Most of the accidents that occur with explosives can be prevented. The safest methods today of handling and using explosives cannot necessarily and unalterably be the method of tomorrow, and their discussion is beyond the scope of this report. Among the many important services given by explosives manufacturers to users of explosives is the sending of their technically trained men into the field to show users the safest and most efficient methods. This same procedure would, if followed by the safety departments of the State and the insurance carriers, be very effective. The work should consist of education in safer practices for all methods of blasting, and be of such a nature as to be within the grasp of the workmen. The explosives companies, especially the group known as the Institute of Makers of Explosives, the State safety department, the National Safety Council, and the United States Bureau of Mines, all issue bulletins describing safe and dangerous practices; what appears to be lacking is a definite and effective educational system at our mines.

An unbiased review of this literature will reveal the fact that electric blasting methods are safer than methods using fuse and detonators.

Some of the common reasons for not using electric blasting that are advanced by ore miners, partly justifiable only because of its being an innovation, are as follows:

1. Electric blasting is not believed to be feasible in stopes and in general ore-mining practice.

2. There are no examples of such practices in metal mines.

3. The copper wires will interfere with gold recovery in milling practice (California).

4. There is a lack of knowledge of safe methods, and no one with experience to instruct in the use of electric blasting methods.

5. Comparisons are not indicative in metal mines because the number of holes blasted under each system is not known.

6. Prejudice exists against electric caps based on a lack of familiarity with their use.

7. A fear exists that they are dangerous to handle.

8. A belief prevails that an electric detonator would cost more than the combination of fuse and cap.

9. Electric blasting involves a costly system of wiring and complicated electric devices.

10. Fuse blasting is simpler for the individual miner.

11. Fuse blasting is safer than poorly installed electrical blasting systems.

12. Electric blasting is not believed to be feasible for bulldozing.

Explosives Records in Anthracite Mining Applicable to Ore Mining

There are no published studies of any importance on ore-mining practice which compare blasting by fuse and detonators with electrical blasting methods. However, a study has been made of the Pennsylvania anthracite blasting practice, by S. P. Howell[13] of the United States Bureau of Mines, which is a valuable experience record for any one interested in explosives accidents and their prevention. The accidents discussed and the conditions under which they occurred have many counterparts in the ore-mining field. This is particularly evident when it is realized that anthracite is hard, blasting is predominantly off the solid in the sense that cutting or shearing is infrequently done, and considerable rock work is done for mining purposes; in fact, anthracite mining has numerous features and hazards analogous to those encountered in metal mining, even the fact that anthracite dust is not explosive. For this reason much more explosive is used per unit of anthracite production than of bituminous coal produced.

Howell enumerates explosives accidents under 17 different classes and details circumstances accompanying 299 fatal and 386 nonfatal accidents involving an estimated firing of 161,265,750 shots, of which 4.4 percent were fired with squibs, 53.5 percent with fuse, and 42.1 percent with electricity. It was found that the greater number of accidents involving explosives were caused by premature shots. In the order of their frequency as to the method of firing used, it was found that when using squibs the rate was 12.2 accidents per million shots, 2.7 accidents per million fuse shots, and 1.9 accidents per million electrically fired shots. Howell says, "Unquestionably, electrical firing of shots is by far the safest method."

In reference to practice in the western metal-mining field, Manning and Soule[14] described a successful electrical blasting system at the Morenci mine

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13 Howell, S. P., Explosives Accidents in the Anthracite Mines of Pennsylvania, 1923-1927: Bull. 326, Bureau of Mines, 1931, 93 pp.

14 Soule, Thomas, and Williams, W. J., How Can Blasting Accidents Be Eliminated?: Address before the California Safety Society, June, 1932.
Soule, Thomas, Blasting with the Inert Primer: Address before the California Safety Society, July, 1932.

of the Phelps Dodge Corporation, Morenci, Ariz. particularly mentioning the use of the inert primer. It is recognized that a careful analysis of actual blasting accidents is of extreme value. Such an analysis[15] has been made of the Morenci operation noting misfires in particular. A careful investigation was made of the results of shooting a total of more than 329,500 shots electrically with inert primers; no misfires could be traced to the use of 152,700 instantaneous electric detonators. The remaining 176,800 detonators were of delay electric types, and of this number there was not a misfire that could be traced to the inert primer where the holes were charged properly.

Electric blasting systems are by no means free from hazards, and unless properly understood, installed, and operated, they can be dangerous. Electric blasting is now required by all companies in Arizona metal mines for long raises and shaft work. It is required in Ontario in all shafts and steep raises. One large Arizona mine uses it exclusively for all classes of work, and others are seriously considering its adoption. The average miner, regardless of nationality, can be taught to blast electrically in a safe and efficient manner, and there is absolutely no good reason why California's mines should not use electrical blasting throughout and be made far safer than they now are.

ACKNOWLEDGMENTS

This report is made possible by the cooperation of the State Compensation Insurance Fund organization and the Safety Department of the Industrial Accident Commission of the State of California. Special acknowledgment is made to C. H. Fry, superintendent of safety, and F. L. Lowell, chief mining engineer, of the Safety Department of the Industrial Accident Commission, to Frank J. Creede, manager, and J. J. Gallagher, secretary, of the State Compensation Insurance Fund, and to the various explosives manufacturers whose data have been drawn upon.

15 See references in footnote 14.

I. C. 6726

June, 1933

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

SAFETY PRACTICES IN TUNNELING OPERATIONS AT THE HETCH HETCHY WATER-SUPPLY PROJECT, CITY AND COUNTY OF SAN FRANCISCO, CALIF.

BY

S. H. Ash and C. R. Rankin

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SAFETY PRACTICES IN TUNNELING OPERATIONS AT THE HETCH HETCHY WATER-SUPPLY PROJECT, CITY AND COUNTY OF SAN FRANCISCO, CALIF.[1]

By S. H. Ash[2] and C. R. Rankin[3]

PURPOSE OF REPORT

Tunnel driving is recognized as one of the most hazardous of occupations; this is a reflection of the high accident severity rates and resultant high compensation and similar costs in this industry. Most tunnels are driven by contractors who bid for the jobs, taking into consideration the prevailing industrial insurance rates at the time the bids are placed. After the bid is secured the contractor then attempts to complete the job at the minimum cost to himself. In a sense the accident-cost item is fixed for at least one year when a policy is secured. Time is generally an important factor not only for the contractor, but also for the insurance carrier and the assured. If the assured represents a private interest, the cost burden usually is ultimately placed on the general public which avails itself of the service contemplated; if the project is a governmental enterprise the cost burden unquestionably falls upon the general public involved and on the general tax payer. For these reasons it is of vital interest to the public that accidents not only be kept at a minimum as a humanitarian objective, but also because the accident-cost burden, too frequently not even realized and considered to the degree that it should be, can assume proportions that materially affect the cost of the enterprise.

The basic industrial insurance rate for tunneling in California in 1932 contemplated an earned premium of 12.12 percent of the labor pay roll. The State has been particularly engaged in large construction enterprises, some of which involve high accident cost rates, notably structural-steel bridge building, which contemplates an earned premium of 19.65 percent of the labor pay roll. When this figure is compared with the much lower rates that apply to other industries it is apparent that a heavy toll in human wreckage is taken by such engineering projects, which must be viewed as monuments to human suffering as well as to human achievement.

This report purposes to show the methods followed at the Hetch Hetchy water-supply project, whereby accidents and their resultant suffering and cost have been minimized because safety has been given a prominent place. Costs in this particular project which will ultimately be paid by the public at large in the district served have been particularly emphasized. The data given are based upon the experience in the Hetch Hetchy tunnels constructed by the forces of the city and county of San Francisco, and cover the period from October 1, 1920, to September 30, 1931, inclusive, during which the compensation insurance item of the industrial insurance cost has been carried with the State compensation insurance fund. The data given in the report should have value to those interested in the relationship of tunneling operations to accident occurrence and accident costs.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6726."

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The tunneling was done through ground that is representative of strata of almost any conceivable type, from the hard, massive granites of the high Sierras, through the metamorphics of the foothills, to the exceedingly gassy, water-laden, heavy, and highly fractured ground of the Coast Range.

The management of the enterprise from its beginning was under the direct charge of M. M. O'Shaughnessy, city engineer, assisted by L. T. McAfee, chief assistant engineer, and P. J. Ost, electrical engineer. The field work was done under the supervision of C. R. Rankin, construction engineer, and his corps of assistants.

ACKNOWLEDGMENTS

In the preparation of this report the writers are indebted to M. M. O'Shaughnessy, city engineer, and L. T. McAfee, chief assistant on public utilities, whose permission and assistance made the report possible. They also acknowledge the assistance and cooperation of C. H. Fry, superintendent of safety, and Fred L. Lowell, chief mining engineer, of the Industrial Accident Commission; of Frank J. Creede, manager; of J. J. Gallagher, secretary of the State Compensation Insurance Fund, in furnishing data relative to accident experience and accident costs; and of A. J. Wehner, safety engineer at the Coast Range division.

THE HETCH HETCHY WATER-SUPPLY PROJECT

The Hetch Hetchy Water-Supply Project of the city and county of San Francisco had its beginning in 1901, but nothing could be definitely done until 1908, when a bond issue was voted which provided for the purchase of watershed lands and water rights. Actual construction work began in July, 1914. The burden of responsibility of carrying to its completion a project, the benefits of which cannot be measured at this time nor will be realized until long after its labors are forgotten, has been largely the task of M. M. O'Shaughnessy, city engineer.

The project is expected ultimately to furnish about 400 million gallons of water daily to 4,000,000 people in the San Francisco peninsula and to develop about 250,000 hydroelectric horsepower for general use for such purposes as household lighting, heating, railway operation, and manufacturing.

The melting snows of the Sierra Nevada are caught and stored in two reservoirs, Hetch Hetchy and Lake Eleanor. Water released from Hetch Hetchy flows 12 miles down the Tuolumne Canyon to unite with the water from Lake Eleanor at Early Intake. Here a dam turns the water into an aqueduct which will conduct it 155 miles to San Francisco, all by gravity - no pumping, and nowhere expose it to contamination. The aqueduct will consist of 82 miles of tunnel 10 feet 3 inches in diameter, and 73 miles of large steel pipes. Hydroelectric power is developed by dropping the water from the high mountain levels to the lower levels. On May 1, 1932, there remained but 4.74 miles, or 0.58 percent, of the tunnel to be driven.

It is beyond the scope of this report even briefly to describe the many problems and details of construction of this notable project.

Its final completion is now a matter of months. A complete description of the problems and progress of the work is contained in the various reports[4] [5] [6] of the Bureau of Engineer-

4 — O'Shaughnessy, M. M., Reports of the Bureau of Engineering, Department of Public Works, City and County of San Francisco, 1908—1931, inc.

5 - O'Shaughnessy, M. M., Hetch Hetchy Water Supply: Bureau of Public Works Bull , Oct. 1925, 47 pp.

6 — O'Shaughnessy, M. M., Hetch Hetchy Project: Apr. 1930, 8 pp.

ing of the City and County of San Francisco. The construction work[7] has been described in part, and recent safety problems[8] [9] [10] and practices emphasized in some technical publications.

Particular interest is manifest at this time in costs, including accident costs at all enterprises and particularly those involving public expenditures. If the basic industrial insurance rate for tunneling can be taken as a criterion of the real hazards in the industry, then tunneling is a more hazardous occupation than mining, as it carries a higher rate; hence, those who are interested in tunneling operations should try by a careful study to ascertain why accident-cost rates for tunneling are higher than for mining when mining is largely tunneling and is often carried on under conditions more inherently hazardous than in actual tunnel construction.

Some hazards of mining which have been encountered in tunneling at the Hetch Hetchy project have received much publicity; the Coast Range division in particular has encountered a methane-gas occurrence in these tunnels such as undoubtedly has never been encountered elsewhere over such distances in tunneling operations in the history of the nation. The methane released per square foot of area opened equals and in some sections exceeds that found in the gassy coal mines of our country. The safety measures that have been adopted, including those for safeguarding against the explosive gas, are not usually found to a similar degree in the majority of our coal mines; hence a brief description is given of the tunnels in the project.

HETCH HETCHY TUNNELS

Aqueduct Tunnel, Mountain Division

The mountain division of the aqueduct, which embraces the Early Intake diversion dam and the aqueduct tunnel to Priest Reservoir, has been conveying an average flow of 735 second-feet of 475 million gallons daily since August 14, 1925, when the Moccasin power plant started generating electrical energy. The tunnel, starting at an elevation of 2,326 feet, is 18.8 miles long and ends at Priest Reservoir at an elevation of 2,170 feet. The easterly 7.5 miles, in hard granodiorite, is 13 feet 4 inches by 13 feet 6 inches in width and height and is not lined. The balance (11.3 miles) is lined with concrete to a horseshoe section of 10 feet 3 inches diameter. Rock encountered consisted of diorite, quartzite, slate, and amphibole schist. The tunnel has a grade of 8 feet per mile. Two shafts were constructed totaling 1,432 feet in depth.

Excavation was prosecuted from 12 working faces; the first heading was begun on July 7, 1917, and the last connection in tunnel driving was made on November 26, 1923. Tunnel lining (60,630 feet) was begun on March 20, 1923, and was completed in May, 1925.

Aqueduct, Foothill Division

The foothill division of the aqueduct is 16.7 miles in length and extends from the tail race of the Moccasin power house (elevation 881 feet) to the Oakdale portal at the eastern

7 – Engineering News–Record, Construction Progress on Hetch Hetchy Aqueduct: Vol. 100, 1928, pp. 614–616; Progress on Hetch Hetchy Water–Supply Tunnels: Vol. 102, 1929, p. 171; Hetch Hetchy Tunnel Construction: Vol. 106, 1931, pp. 96–100.

8 – Harrington, D., Progress in Metal–Mine Ventilation in 1930: Inf. Circ. 6469, Bureau of Mines, 1931, pp. 5–12.

9 – Jarrett, S. M., Safety Progress on the Hetch Hetchy Aqueduct: Nat. Safety News, March, 1932, p. 21.

10– Ash, S. H., and Rankin, J. H., Permissible Electric Cap Lamps and Ventilation in Certain California Mines and Water–Tunnel Construction: Bull. 359, Bureau of Mines, 1932, 36 pp.

edge of the San Joaquin Valley (elevation 747 feet), which is the beginning of the 47.5-mile pipe line connecting with the Tesla portal, the beginning of the Coast Range tunnels.

The foothill tunnels were started in February 1926. All work was done by day labor with city forces until the fall of 1926, when two contracts were awarded for carrying out certain portions of the work. The tunneling and lining operations were completed in October 1929. The total length of the tunnels in this division is 83,663 feet (as excavated) or 15.85 miles, of which 36,887 feet were excavated by contractors and 46,776 feet by city forces under the direction of the city engineer. Of the 83,663 feet of tunnel driven, 43,176 feet were lined and 40,487 feet unlined. Of the 43,176 feet lined, 30,732 feet were lined by city forces and 12,444 feet by contractors. The accident experience of the contractors is not available and is not included in this report.

San Francisco Bay Division

The San Francisco bay division is a section of the aqueduct, 21 miles long, completed from Irvington at the west portal of the Coast Range tunnels, and brings additional water from Alameda County pending completion of the Coast Range tunnels. Included in this division is the Pulgas Tunnel, the west end of the development, near Crystal Springs Reservoir. This tunnel was completed in May 1924, is 8,676 feet long, has a diameter of 10 feet 3 inches, of horseshoe section, and is concrete lined. All but 240 feet required timbering. The work was done by contract, and the accident-cost experience is not included in this report.

Aqueduct Tunnel, Coast Range Division

The Coast Range division of the aqueduct was just getting under way at the beginning of 1928, and is to extend from the Tesla portal at an elevation of 399 feet, to the Irvington portal, at an elevation of 316 feet; this is the remaining intervening section to be completed between the San Joaquin Valley pipe line and the San Francisco Bay division. The valley of Alameda Creek, which will be crossed by a pipe line 0.6 mile in length, divides the Coast Range tunnel into two sections, which are 25.1 miles and 3.4 miles long. The short section of 3.4 miles on the Irvington end has been driven, and the work of concrete-lining its entire length is nearing completion.

The 25.1-mile tunnel is worked from two portals and five shafts, being thus divided into six sections. On March 4, 1931, excavation was completed in the Tesla portal-Thomas shaft section, a distance of 4.4 miles, and the concrete lining for the entire distance is nearing completion. The longest and most difficult section lies between the Mocho shaft and Mitchell shaft, a distance of 5.2 miles, which should be completed by the close of 1933. Excavation between the Mocho and Valle shafts was completed May 13, 1932.

Of the total 151,227 feet (28.64 miles revised distance) of tunnel in the Coast Range division, 126,196 feet had been driven by May 1, 1932, which left 25,031 feet, or 4.74 miles only, to be driven, - only 16.55 percent of the total length of the Coast Range tunnels. During the fiscal year July 1, 1930, to June 30, 1931, a distance of 7.2 miles was driven. The driving of the Coast Range tunnels has been done entirely by San Francisco City forces.

Extraordinarily difficult conditions have been encountered in the driving of the Coast Range tunnels, which are to be concrete-lined for their entire length. These difficulties have been numerous and of varied character: Swelling ground has been encountered that requires considerable retimbering until a type of concreting designated as guniting has been done; quicksand and water have contributed their share in retarding normal progress in places; the unusual prevalence of large amounts of the explosive gas methane, which causes so much trouble in coal mines, has necessitated the adoption of such explosion preventive

measures as regards equipment, ventilation, and supervision as are seldom found even in gassy coal mines and not at all in the usual tunneling operations; and occurrences of hydrogen sulphide gas have been encountered, but have been successfully handled by increased ventilation and care. That the safety precautions adopted have accomplished results is unquestionably shown by a study of the statistics included in this report. A disastrous explosion such as could, and very likely would, result if adequate precautions were not taken, would not only cause loss of life, but probably also cause enormous property damage in the section involved. To avert these possible catastrophes, too often accepted as hazards of the industry, the city engineers, the State Industrial Accident Commission, and engineers of the United States Bureau of Mines have cooperated in a constructive endeavor to safeguard the workmen, which has been eminently successful. It is of interest to know the nature and results of the safety practices that are in effect at present with the sole purpose of preventing accidents and disasters. The particular data reported here reflect the experience in the Coast Range tunnels. It is with the tunneling operations of this section of the aqueduct that the remaining descriptive matter in this report deals primarily.

OPERATING FEATURES, COAST RANGE DIVISION

Size of Operations

Because of proximity to the Bay region around San Francisco the operations of the Coast Range division have assisted materially in relieving unfavorable conditions in this area during the present depression by maintaining a large pay roll and demand for materials. The pay roll in the tunneling operations alone was over two and a half million dollars in 1931 and was approximately two million dollars in 1930. Coming at this time this unquestionably has contributed much to the welfare of the region. During 1929, the latest available year, the pay roll on the Hetch Hetchy project was 87.8 percent of the entire State pay roll for tunneling work that was insured by insurance carriers.

Excavation progress as of May 1, 1932, is shown in Table 1.

The number of employees in the tunneling classification during 1931 averaged 1,520 three-hundred-day workers; this classification represents approximately 80 percent of the total number of persons employed at the various camps. The remaining 20 percent employed were in other classifications on the surface and are not included in this report.

The degree of work carried out in the different headings has varied with conditions. A working day of 8 hours has been in effect at this enterprise and material has been hauled on all three shifts normally worked underground. The average tunnel crew in each heading for one shift consists of one superintendent, who looks after the whole camp, one shift boss, one mucking machine operator, one motorman, one nipper, four miners, three chuck tenders, four muckers, one skip tender, one station tender, one pumpman, and one fire boss. In addition, there may be other miners engaged in retimbering, two or more men guniting in sections, and men who work on any shift, such as a trackman, a pipeman, and a track foreman. A night walker replaces the superintendent on the afternoon and night shifts. Members of the engineering staff are constantly employed underground in some section or other.

Table 1.- Main aqueduct, Coast Range division excavation progress, May 1, 1932

Heading		Station, April 1	Station, May 1	Progress for month	Progress to date	Total remaining
Tesla	173+45					
	West	312+49	312+49	-	13,904	-
	(East	312+49	312+49	-	9,406	
Thomas	(					
	(406+55					
	(West	494+98	498+94	396	9,239)	8,311
					)	
	(East	584+11	579+36	475	4,894)	
Mitchell	(					
	(628+30					
	(West	696+74	698+58	184	7,028)	10,965
					)	
	(East	807+43	804+22	321	9,973)	
Mocho	(					
	(903+95					
	(West	979+76	982+17	241	7,822)	179
					)	
	(East	986+21	983+96	225	7,302)	
Valle	(					
	(1056+98					
	(West	1146+98	1148+89	191	9,191)	4,579
					)	
	(East	1196+66	1194+68	198	13,102)	
Indian Creek	(1325+70					
	(West	1356+93	1356+93	-	3,123)	997
					)	
	East)	1366+90	1366+90	-	13,020)	
	)					
Alameda Creek	1497+10)					
	1528+29)					
	West)	1616+35	1616+35	-	8,806	
	East)	1616+35	1616+35	-	9,386	
Irvington	1710+21					
Total				2,231	126,196	25,031
					151,227	
Top drift, Indian W. (1365+82)		1360+11	1361+86	175	-	-

Surface Plant and Equipment

Slow-burning wood construction throughout for headframes is used at all shafts. Fire protection is provided at these locations by hose, hydrant, and sprinkling systems. Chemical extinguishers are also provided. No regular fire drill is held.

Gasoline locomotives are used on the surface for handling muck cars. Double-drum, electrically driven hoisting engines with a maximum speed of 600 feet per minute are used; overwind and overspeed equipment are not provided. The surface electric voltage is 440 volts a.c.; the energy is obtained from the City-owned power plant, transmitted at 22,000 volts, and stepped down to 440 volts by transformer banks efficiently installed near shaft portals. These enclosures are kept locked. Electricity is used throughout for surface lighting.

A washhouse is provided at camps, with combination shower house and hot and cold water, together with lavatories and sanitary toilets.

The workmen are housed in 5-man bunk houses. Drinking water is supplied from springs and is regularly analyzed. Cookhouses are efficiently maintained at all camps.

The men are checked into and out of the tunnels by means of identification tags taken from a rack by the men on each shift, and time cards which are taken from a rack and marked by shift bosses. Both identification tags and time cards are returned at the end of each shift.

Underground Equipment and Haulage

Man-cages of Joshua Hendy design are used in the shafts. These cages are equipped with safety devices, and drop tests are performed regularly at 2-week intervals; the cages have steel-bonnet protection and are cleaned and oiled daily.

Ladders and safety gates are maintained in the shafts, which are efficiently concrete lined. The ropes are inspected at regular intervals.

Rails of 30, 40, and 50-pound weight are used for the underground haulage system, which has a track gage of 24 inches.

Forty-one storage-battery locomotives of General Electric, Westinghouse, and Mancha manufacture are used underground. Twenty-seven of these are of permissible types operating at 80 volts for use in the tunnels known to give off explosive gas. Eighteen are of 5-ton weight and 23 of 4-ton weight.

The cars in use are of 2-yard Koppel-steel and 3-yard Western dump - wood body, designs. There are no grade hazards.

The tunnels are not lighted with incandescent lights in the gassy sections, as this is known to be a serious potential explosion hazard.

Separate man-trips are provided and men are not permitted to ride on loaded trips; workmen are not permitted to ride on bumpers, nor to make flying switches. Tail lights of the permissible electric storage battery type are used on all trips.

Explosionproof motors of permissible types are used for pumping units in some headings where methane gas is present; other pumping units in the headings are of duplex design and are compressed-air driven. All underground electric equipment is operated in intake air.

Compressed air operated mechanical loaders are used, one in each heading, and a spare is carried for contingencies. The loaders are of Conway and Myers-Whaley manufacture. There have been no fatal accidents directly assignable to these loaders. The number of men on a loader crew varies, but the usual heading crew will consist of 12 men.

All mechanical electrical equipment is inspected daily and an effort made to maintain the permissibility of the permissible equipment.

The Coast Range tunnels have been timbered throughout, and a portion of timber has been supplanted in places with guniting. The type of timbering has varied, but the typical set is a 6-segment arch with plumb posts. The distance between sets depends on the length of the round broken and the nature of the ground encountered. Systematic timbering rules are printed and a continual effort is made to carry out the orders.

Explosives

The use of explosives is one of the major means of operation in tunnel driving. From the point of view of safety, it is probable that unsafe blasting practices offer the greatest hazards in the average tunnel for large groups of the tunnel workmen. The use of explosives on the Hetch Hetchy project represents a welcome transition from the unsafe crowding practices usually obtaining in contracting work with fuse and caps, to the up-to-date, much safer method of using permissible explosives and electric blasting. There is probably no better example of what can be accomplished in safeguarding blasting practice than will be found on the Hetch Hetchy project.

Between the years 1920 and 1923 explosives with fuse and detonators caused four explosions, which cost the lives of 11 persons, and injured 10 others; these explosions occurred during the driving of approximately 85,000 feet of tunnel. Following this period, electric blasting was adopted, and but one explosive accident has occurred, one life being lost while tamping a primer. During this latter period, and up to May 1, 1932, there have been 209,859 feet of tunnel driven. Unquestionably, the substitution of electric blasting has been largely responsible for the great improvement in safety achieved in the latter period.

The number of explosive accidents in this tunnel work not involving a loss of life is not known. The use of the inert primer[11] in tunneling operations would unquestionably aid in the prevention of explosives accidents in connection with missed shots, striking primers, removing charges, transportation, etc., – the oauses of most explosives accidents in tunneling and metal-mine operations.

Gelatin, Colites, and Herco-Gel explosives have been used in these operations since 1930. Electric blasting caps of No. 8 strength are employed.

The eight surface magazines at the respective camps are constructed to conform to California laws of distances and are bulletproof.

The explosives are hauled into the tunnels by the motorman; 2-compartment, insulated powder cars are provided for the explosives and detonators, which are kept separate.

A maximum of 100 pounds of explosive is stored underground in locked wooden boxes; this explosive is for single-shot plugging done with permissible explosive and permissible single-shot blasting units. Explosive sufficient for a round of about 40 holes is brought into a tunnel direct from the surface.

Explosives and detonators are not handled by the same man at the same time. The miners and shift boss load the holes. Cartridges are used and air spacing is guarded against. The cartridges are slit and the explosives carefully rammed. Clay stemming is used and a wooden-type bar is used in tamping.

Rounds of 20 to 40 holes are drilled with compressed-air drills using water. Four to seven sticks (1-1/8 by 8 inches) of permissible explosive are used per hole. From 5 to 9 feet are "pulled," depending on the ground.

Shooting is done by a fire boss who has satisfied the Industrial Accident Commission as to his ability to handle gassy conditions; one fire boss is employed on each shift. From 1 to 40 shots are fired at one time, delay-action exploders being used.

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11 – Manning, R. I. C., and Soule, Thomas, Electrical Blasting Practice at the Morenci mines of the Phelps Dodge Corporation, Morenci, Arizona: The Explosives Eng., June, 1931, and July, 1931.

Power blasting is used for the main rounds, a special blasting switch[12] being employed for this purpose. The locked switch is operated by the fire boss from the shaft crosscuts with all men removed about 5,000 feet or more from the face. Permissible single-shot blasting units operated by the fire boss are used for plugging.

Misfires were common prior to the adoption of electrio blasting, and are assigned as the cause of all previous explosives accidents, costing 11 lives; no fatalities have been caused by misfires since the adoption of electric blasting, though two nonfatal accidents occurred when unexploded primers used for plugging were struck when picking for posts. Inert primers very likely would prevent this type of accident.

In case of known misfires, which are examined and reported, a new primer is used and the hole is again fired. The legs of all electric detonators are short-circuited and the "short" is removed before loading. No record of misfires is available, but it is known that they are rare and appear to be a result of series-blasting for plug shots.

Electricity Underground

All high-tension electrical energy is transmitted underground by armored cables at 2,200 volts a.c. The low-tension voltage is 110 and 440 volts a.c.; no direct current is used underground except at storage battery charging stations. The motor-generator sets are placed on separate air splits, in concrete sections. Transformers are set up in concrete stations, and the fire hazard is kept at a minimum.

Wiring is carried in conduit and separate power switches and circuits are used. Electricity, except under permissible conditions, is kept out of all gassy tunnels, and incandescent lighting is used at the shaft stations only in strictly fresh air.

The electrical system and equipment are inspected monthly by the electrical engineers and a report is made to the construction engineer. There have been no electrical contact accidents which have resulted fatally or seriously.

GENERAL SAFETY PRACTICES AND ACCIDENT PREVENTION

During the sinking of the Mocho shaft in 1928, it early became evident that the hazards from the explosive gas methane and the toxic hydrogen sulphide gas should receive serious consideration; explosions of methane have been by no means uncommon in tunnel operations[13] in the Coast Range Mountains when a certain formation, the Franciscan, is penetrated.

Adequate ventilation of workings has been given a prominent place in the activities of the Coast Range division, and this phase of operation of these tunnels should be of value in indicating safe yet efficient ventilation and operating measures in future tunnel driving in gassy strata. The possibility of occurrence of methane is not to be considered a "chance" hazard in any Coast Range tunneling operations in the Franciscan or Cretaceous formations; it should be recognized as a "real" or at least a probable hazard in this district by the State, insurance carrier, contracting party and contractor, and prior to engaging in tunneling operations in this region steps should be taken to provide suitable types of equipment to avert hazards before explosions or fires occur.

Ventilation and Gas

Ventilation of the Coast Range tunnels is by means of Roots rotary-type, positive-

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12 - See footnote 10.

13 - Harrington, D., Work cited, p. 9, Progress in Metal-Mine Ventilation in 1930: Inf. Circ. 6469, Bureau of Mines, 1931, p. 9.

pressure blowers of three sizes - 5, 5-1/2, and 7 - installed on the surface in fireproof,[14] galvanized-iron buildings placed 30 to 50 feet from the shaft openings. Reserve motive power is available.

Metal pipe of 16 gage and 24-inch diameter is used, and the blowers are operated as suction units. Prior to operating on an exhaust system with the relatively high pressures obtainable with this equipment, it was believed that the pipes would collapse; however, as precautionary measures required an exhaust system, tests[15] were made with a stiffener band placed on the outside of the metal tubing to prevent collapse or deformation. These bands have accomplished this purpose. Obviously, if there is no deformation there could be no collapse. To insure against collapse, pressure readings are taken with a manometer and a record of them is kept.

In the Indian Creek tunnel 6,078 cubic feet of air is circulated through 13,500 feet of 24-inch pipe at a vacuum of 1.85 pounds per square inch at the fan on suction; this was not believed possible of achievement until it was actually done.

If there are any interruptions in the running of the fan at the gassy tunnels, notification is made from the surface and the men are sent out, or if the fire boss or shift boss first notes the interruption he orders the men out of the mine.

Under the heading of "tunnel ventilation" Harrington[16] discusses the ventilation of the Coast Range division of the Hetch Hetchy project and the accident-prevention measures which are being taken.

It has been frequently found, not only in the Coast Range tunnels of the Hetch Hetchy project, but also in other tunnels in this vicinity, that methane gas may be liberated suddenly; it is a constant hazard. In one tunnel alone, 116,900 cubic feet of pure methane was liberated in 24 hours in some 9,783 linear feet of tunnel.[17]

To provide adequate protection and prevent the accumulations of dangerous quantities of explosive gas, certified fire bosses are employed on each shift. Permissible flame safety lamps are kept in each heading, but on account of the danger always accompanying the use of flame safety lamps if they are not properly assembled and handled, their number is kept at a minimum.

To inspect for gas, which is done continuously during the shift, the fire bosses use U.C.C. methane indicators. The superiority and safety of these devices for accurate and efficient inspection has been emphatically demonstrated in connection with the driving and handling of these tunnels. A detailed report is made by each fire boss at the close of each shift; a copy is sent to the city engineer's office and a copy to the State Industrial Accident Commission.

It is probable that methane indicators find their largest use in the construction field, and that the cost items for them in that industry would be high as compared with the same items in the mining industry; this also applies to permissible electric cap lamps.

The cost of maintaining 12 methane indicators in operation during 18 months was $980 for repairs, $2,208 for 830 filaments, and $468 for battery cost, replacement, and overhaul - in all, $3,656.

These indicators have been in constant use on the Hetch Hetchy project since May, 1930, at each camp. Fire bosses trained in their use patrol the working places on each shift and record readings at the headings at least hourly. The indicator quickly notes percentages of methane from 0.10 percent to explosive mixtures.

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14 - See footnote 10.

15 - See footnote 10.

16 - Harrington, D., Work cited, pp. 5-11.

17 - Ash, S. H. and Rankin, J. H., Work cited, p. 25.

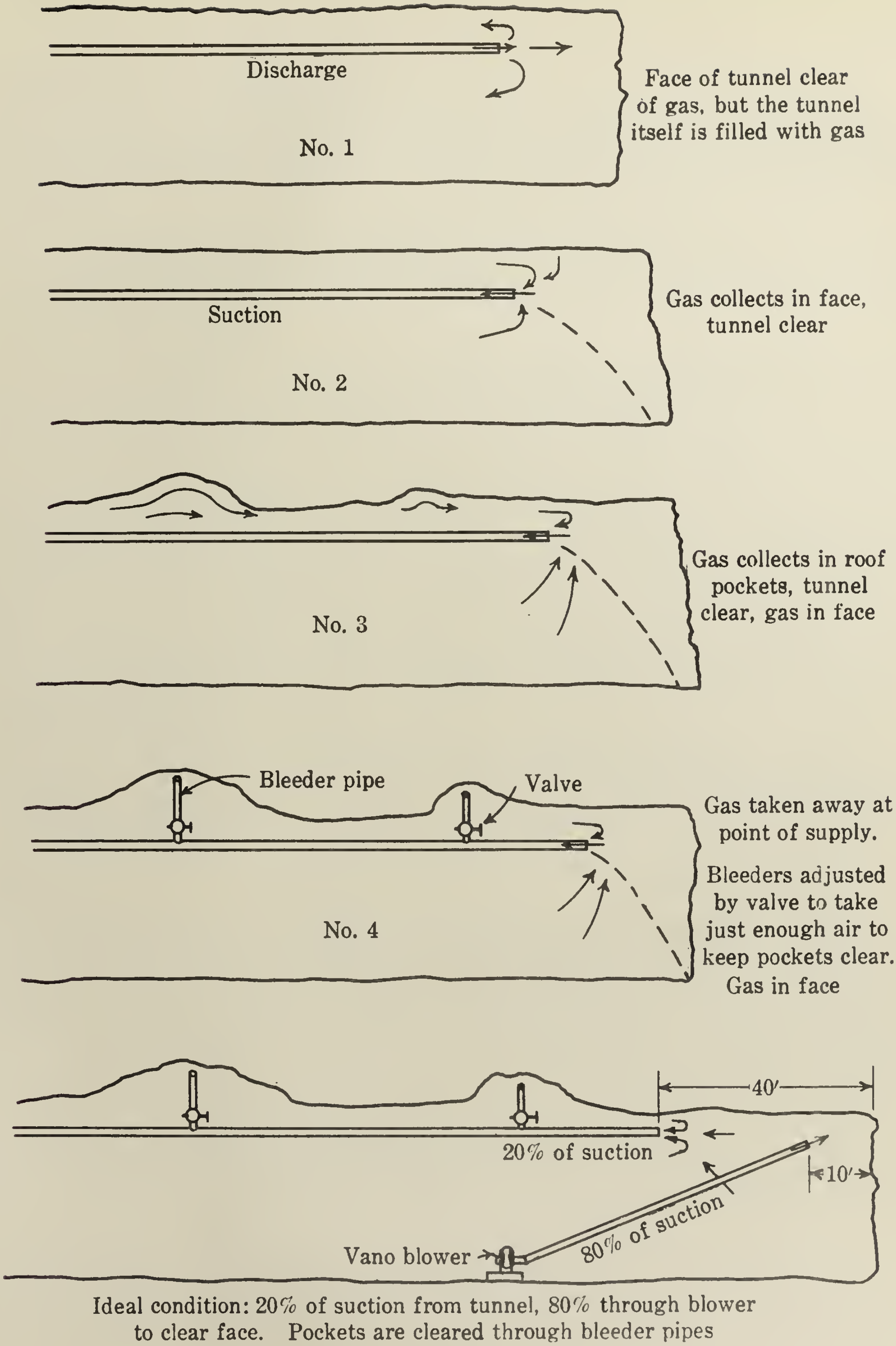

Figure 1.- Method of using blower pipes in tunnel ventilation.

With the U.C.C. indicator it is possible quickly to determine the percentage of methane along the length of the tunnel and readily define the depth of gas strata and percentage curve. This would be impossible with a flame safety lamp, a slow and questionable determination with a Burrell methane indicator, and an unduly costly and delayed process by any sampling method.

The average life of filaments depends to a large extent on the skill of the operator, as well as on the percentages of gas encountered. Early experience was 79 to 82 hours use per filament, but this has been increased considerably since the gases have been piped to the main exhaust line.

The zero point of the U.C.C. indicator must be checked at least every hour in fresh air, as it is found to vary much at times if held continually in gas-impregnated air. The checking can be done by going to the surface, or where compressed air is in use, by placing the instrument in fresh air from the compressed air line at a favorable point.

It was found that special attention must be paid to the batteries to get the best results. Charging was never done at a greater rate than 1 ampere, as it was found that in this way the cells do not overheat and yet secure the full benefit of the charge. The charging time averaged about 8 hours, and about once a week an overcharge was given of about 16 hours at not more than 3/4 ampere-hour. The batteries should be given a water wash and a new solution at least every three months if in constant use. The solution in both cells was checked every day to see that it was not only at the same level but also gave the same reading. With careful treatment of the terminals, little trouble with the batteries was experienced after starting to handle them as described, although previously trouble was more or less constantly experienced.

Experience and study with the U.C.C. detector indicate that it can fill a valuable place in practical gas analysis and in accident prevention in mines and tunnels. However, the following improvements appear to be desirable: A smaller gas chamber around the filament, a better method of zero adjustment, a simplified battery, and adaptability to any permissible electric cap-lamp battery.

The sequence of ventilation changes that were made in the tunnel ventilation is shown in Figure 1.

The amount of fresh air intaking at the different tunnels varies with conditions as regards gas and temperatures and has ranged from 6,000 to 30,000 cubic feet per minute.

Samples of mine air are collected from time to time and analyzed by the United States Bureau of Mines. The results are studied and desirable changes made in ventilation practice.

Lighting in the gassy tunnels is by means of Edison permissible cap lamps. Incandescent lamps in these gassy tunnels are used at the shaft landings only in fresh air. Probably most of the methane explosions that have occurred in the tunneling operations in gassy strata in California, including Hetch Hetchy, are the result of some failure in the incandescent lighting system used in tunnel illumination; these systems cannot be made foolproof. The permissible electric cap lamps remove the explosion hazard and give dependable, effective lighting, and they should be more generally used in all kinds of underground mining operations.[18]

The following data (Table 2) relative to the costs of such a lighting system are of interest. Aside from the safety factor, which was the determining influence in their selection in this instance, they are efficient and tend to reduce accidents. The information in Table 2 covers the use of Edison permissible cap lamps from July, 1930, to April 15, 1932. The scattered locations make the item of attendant's time unusually high as it has to be divided between 8 camps for less than a total of 1,000 lamps. In most situations an attend-

18 — See footnote 10.

ant's time would not be wholly occupied with the lamps, or he could handle a larger number. The compilation is at least fairly reliable, is conservative, and represents maximum rather than minimum figures as to costs.

Table 2.- Costs of operating Edison electric cap lamps at the Hetch Hetchy project, July 1, 1930, to April 15, 1932

	Total cost	Unit cost, cents per man-day
979 Edison cap lamps complete, Model H	$13,216	-
20 extra batteries	213	-
8 charging racks	1,120	-
8 Tungar chargers	440	-
Total equipment	14,989	1.98
Battery repair parts	1,087	-
Lamp repair parts	6,328	-
9046 bulbs	1,536	-
Total repairs	8,951	1.18
Lamp attendants and electrician's time	25,000	3.31
Power	125	0.016
Grand total	49,065	6.494
Man-days of labor performed		755,664

In addition to the use of permissible electric cap lamps a storage-battery flood light[19] was developed by the city engineer's office and is used at the loading machines.

At irregular intervals the men entering the underground workings are searched for matches and smokers' articles; any one found with such articles is disciplined by discharge. A form is used for recording the results of the search. The California mining laws do not define such an offense as a misdemeanor.

An assistant engineer of the project is employed as a safety engineer, whose duty it is to inspect the tunnels, report concerning the safety conditions, and make suggestions for improvements to the construction engineer. In addition to a verbal report an inspection report form, Figure 2, is filled out.

Supervision

There is probably no factor in safeguarding property and the workmen in any industry of more importance than efficient supervision in a safe working place. Without it there can be no effective safety program and certainly no efficient operation.

Undoubtedly no type of construction work requires more adequate and efficient supervision than tunneling operations; this fact has been kept in the foreground on this project, and an average of only about 12 men is employed under each face boss. The face boss spends most of his time at the face and is not hurried in any visits. Whenever orders are given it is insisted that they be carried out.

19 - See footnote 10.

INSPECTION REPORT

CAMP ____________________ Date __________________

Tunnel heading ________________

Kind of ground ____

Timber alinement ____

State of timber ____

Timber alinement ____

Blower pipe ____

Gas condition ____

Equipment

Air-leakage 4-inch pipe, receivers ________________________________

Line oilers ____

Safety lamps ____

Coppus blowers ____

Fire protection (Camp and tunnel)

Hose and racks ____

Old timber refuse ____

Sanitation

Eating place canvas ____

Miscellaneous

Man cages __________________ Tests _____________________________

Pumps ___________ too many ___________ too few

____
Assistant Engineer

Figure 2.- Inspection report form

When an entire week passes without any industrial accidents in the tunnel work, a special chicken dinner is served to the entire camp; several chicken dinners have been earned since this system was inaugurated.

Special safety orders[20] issued by the Industrial Accident Commission, other working rules, and posted notices are put into effect as is found to be necessary.

Miscellaneous Hazards

To minimize the fire hazard, carbon dioxide extinguishers are provided for all electrical equipment. Other extinguishers are placed at points of hazard. Galvanized-iron buildings only are used on the surface. Oil used underground is stored in concrete stations.

The shafts are lined with concrete and all shaft stations are of concrete construction.

The ventilation is of such nature that direction of air flow is reversible; the system has always been maintained in this condition.

Fire-protection pipe lines are laid throughout the tunnels with water taps at 250- to 500-foot intervals. Three hose stations are maintained underground for each shaft; 2-inch hose in 100-foot lengths, one carbon dioxide fire extinguisher, and six soda extinguishers are provided at these stations. Two-inch water lines and fire hose are provided at each heading and at the shaft crosscut.

The flood hazard in the Coast Range division has been efficiently guarded against in the areas where quicksand and water have presented that hazard to a limited, but for a time, troublesome degree.

The exhaust system of ventilation, together with fairly high rock temperature and high relative humidity, makes working conditions somewhat uncomfortable and means are being taken to protect against high wet-bulb temperatures in lining operations.

To prevent head and foot injuries and to minimize the severity of such injuries as do occur, the men are furnished with hats for head protection and with safety shoes to protect toes and ankles.

Mine-Rescue and First-Aid Practices

On June 8, 1930, an explosion[21] due to methane in the Upper Alameda Creek Tunnel of the old Spring Valley system caused the death of seven persons. Events following this disaster emphasized the importance and necessity of having a sufficient number of persons trained in the use of mine-rescue oxygen breathing apparatus in the Coast Range division of tunnels operating in the same geological strata. Accordingly, a mine-rescue station was equipped at the Hetch Hetchy project, and mine-rescue training began on June 21, 1930, under the supervision of the United States Bureau of Mines. The efficiency of the men trained at this project in this work has been demonstrated in an actual emergency, and the work has been consistently maintained.

The management of this project has recognized the fact that the chance of injury is usually minimized for a man trained in first aid, as he has become "safety conscious." At the request of the city engineer, in March 1931 a program was started to train every workman on the project and to maintain this status. Contests in this work are held from time to time between crews from the different communities, and training is kept in effect.

A mine-rescue station is maintained at Livermore, as it is central. Here are 10 sets of Gibbs oxygen breathing apparatus, an oxygen pump, an inhaler-type oxygen reviving device,

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20 - See footnote 8.

21 - See footnote 10.

120 regenerator charges in stock, and 12 to 18 oxygen tanks. In addition, there are 33 gas masks and a stock of 75 canisters, all in working condition. The apparatus is used monthly and kept in condition for wearing.

The apparatus is in charge of A. J. Wehner, safety engineer, and S. M. Jarrett, first-aid and mine-rescue instructor and relief fire boss.

That the goal of 100 percent first aid training is in sight can be realized when it is recognized that 1,200 persons have been trained in first aid; in addition, 120 have been instructed in mine rescue and 46 in advanced mine rescue and recovery work, and both groups have been certified by the U.S. Bureau of Mines since June 1930. Some of the camps are 100 percent trained in first aid. This group of persons is the largest so trained in any individual mining or tunneling enterprise in California.

In addition to the first-aid or mine rescue certificate furnished by the United States Bureau of Mines, each person completing the training work is given a certificate issued by the project.

Completely equipped canisters, stretchers, and first-aid kits are maintained underground at each tunnel. Similar equipment, including stretchers, blankets, and other supplies is kept in the first-aid training rooms maintained on the surface at each camp. The nearest hospital is at Livermore, which is 9 to 19 miles distant.

Safety Organization

A safety organization is maintained and directed by the construction engineer, assisted by the safety engineer; monthly discussions are held in which the tunnel foremen and fire bosses participate.

A safety committee functions at each tunnel; its duty is not only to suggest safety ideas, but to assist in carrying them out. This committee consists of the camp superintendent and three workmen, the safety inspector acting as secretary. A report of the serious accidents investigated by this committee is made as shown in Figure 3.

METHOD OF REPORTING ACCIDENTS

All accidents are reported by the fire bosses and timekeepers to the safety engineer, who in turn refers them to the construction engineer through the chief clerk. A special clerk in the Livermore office handles all accident claims and submits them to the insurance carrier.

At the Hetch Hetchy project the city provides the medical care, while the compensation insurance item is carried by the State Compensation Insurance Fund.

If the case is one for a hospital or if medical treatment is required the injured person is given a certificate of entry to the hospital (fig. 4) by the clerk. When the workman is ready for work a discharge slip (fig. 5) is given to him which must be presented to the timekeeper, who assigns him to work. These slips, including the form (fig. 6) filled out by the attending doctor, are given to the clerk who attends to the compensation insurance items. Such items are reported on suitable forms.

MEDICAL SERVICE

For many years the city has maintained a hospital equipped to handle all cases, both medical and industrial. Until May 1930 this hospital was at Groveland, near the foothill division of the tunnels. With the object of having a hospital nearer and in the center of operations of the Coast Range division, the city's Livermore hospital was acquired and opened for service on August 25, 1930.

SAFETY FIRST

Camp ______________________

REPORT OF ACCIDENT INVESTIGATION

Name of injured party _____________________________ Badge No. ___________

Work employed at ____

Place of accident _______________________ Date __________ Time _________

Description or cause of accident:

Description of injury:

Was accident caused by carelessness of injured or negligence of some one else?

What rule was violated:

Suggestions for prevention of similar accident:

Discipline recommended (if any)

APPROVED

Investigated by: ______________________________ Superintendent

____
______________________________ Safety Committee.
____

Figure 3.- Accident investigation report form.

Form 176 **CERTIFICATE OF ENTRY TO HETCH HETCHY HOSPITAL** 20M-7-31

LIVERMORE, CALIFORNIA

Camp..Date.......................

Employee's Name..No..............

Occupation..

Employer (Other than H. H. W. S.)..

Industrial ( ) Non-Industrial ( ) Date of Accident or sickness.......................

....

Timekeeper

TO BE FILLED IN AT HOSPITAL

History..

Diagnosis...

First Treatment..

Figure 4.-Certificate of entry to hospital.

Form 177

HETCH HETCHY HOSPITAL

CERTIFICATE OF DISCHARGE

....192........

Construction Engineer:

This will certify that bearer,

....

has been discharged from the Hospital as being in fit condition to return to work.

....

PHYSICIAN

NOTE: PRESENT THIS CERTIFICATE AT GROVELAND OFFICE FOR RETURN TRANSPORTATION

Figure 5.- Certificate of discharge from hospital.

H.H.W.S Form 266 5M-1-31

HETCH HETCHY HOSPITAL

Livermore, Calif. 193

Case No.

Employer

Injured

Date of Injury

1. Final Diagnosis

2. Present Condition

3. Any Permanent Disability?

4. Discharged From Hospital?

5. Returned to Duty

Remarks:

J. P. DEGNAN, M. D.

Figure 6.- Form filled out by attending physician

The main hospital building is 187 feet by 41 feet and contains 31 beds and full equipment. The members of the hospital staff, Dr. John P. Degnan in charge and eight assistants, are subject to call at all times. Three cottages are provided for the hospital staff.

All employees are subject to physical examination at the hospital before entering the service of the city, and each employee is assessed $1 per month for medical service.

Ambulance service is provided, subject to call day or night, for transportation between the eight camps and the hospital.

I.C. 6727
June, 1933.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

FACTORS AND CONDITIONS THAT AID IN ALINEMENT OF PILLAR EXTRACTION LINES IN COAL MINING[1]

By J. N. Geyer[2]

IMPORTANCE OF PILLAR RECOVERY

During the early stages of the mining industry of the United States an apparently inexhaustible supply of high-quality, easily accessible coal made the product cheap and the attendant mining methods wasteful of recoverable coal. Because coal from advance places could be mined with less effort and a large output obtained as the entries advanced, much of the coal left in pillars to support the roof was lost. Squeezes often developed in these mined-over areas, or, if pillars were of sufficient size to prevent squeeze, the roof would be so weakened by long exposure that any pillar recovery was extremely hazardous.

As the value of unmined coal increased, a high recovery of the available coal became necessary. Numerous plans were tried during the period that followed, some of the methods proving successful while others were complete failures or only partly successful. As a better knowledge was gained of the roof strata and its behavior, however, pillar mining became more successful and methods suited to special roof conditions were developed.

In the study of falls of roof in bituminous-coal mines conducted by the Bureau, particular attention is necessarily directed toward the methods of mining used as they affect roof control in pillar extraction. As a result of these studies, several methods for winning the pillar coal and controlling the roof along rib lines under diverse natural and economic conditions have been observed, and it is believed that a brief discussion of these methods and their application will be of material aid in recovering the coal in other mines having similar conditions.

The data used in preparing this paper have been compiled from reports made in connection with the study of falls of roof and coal by Bureau engineers and includes coal mines in western Pennsylvania, northern West Virginia, and a part of southern West Virginia.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6727."

2 Associate mining engineer, U. S. Bureau of Mines, Pittsburgh Experiment Station, Pittsburgh, Pa.

DESCRIPTION OF THE DISTRICTS

In Pennsylvania the studies have been confined to mines in the Pittsburgh bed in Allegheny, Washington, Greene, Westmoreland, and Fayette Counties. Mines studied in West Virginia include those in the Pittsburgh bed in the Panhandle district; the Pittsburgh and Sewickley coals in the Fairmont district; and the Pocahontas No. 3 coal in McDowell County.

Topography

The topography ranges from gently rolling to broken in the Panhandle in West Virginia and the western portion of the field in Pennsylvania. In the eastern part of the field in Pennsylvania and in the Fairmont district in West Virginia the surface is more rugged, the streams having cut deep V-shaped valleys leaving comparatively sharp ridges. In McDowell County the surface is decidedly rugged.

Thickness of cover

The strata overlying the coal ranges from a few feet at outcrop to 1,000 feet in a few places. Covers of 500 feet are common over the Pittsburgh coal in the gas and coke region of Pennsylvania and in the Fairmont district of West Virginia. Over the Sewickley coal, where mined in West Virginia, the cover seldom exceeds 300 feet. In McDowell County, W. Va., the cover often exceeds 500 feet and as much as 1,000 feet has been encountered. The following table gives the range in cover over each group of mines studied:

Cover or overburden on coals in the district studied

State and district	Cover, feet			Number of mines
	Minimum	Maximum	Average	
Pennsylvania:				
Pittsburgh bed -				
Allegheny and northwestern Washington Counties	Outcrop	500	165	8
North-central Washington County	25	800	295	8
Irwin, Greensburg, and Latrobe basins	Outcrop	550	265	12
Eastern Washington and Greene Counties	125	900	430	6
Fayette County	Outcrop	815	388	11
West Virginia:				
Pittsburgh bed -				
Panhandle	Outcrop	730	240	6
Harrison County	Outcrop	400	175	5
Marion and Monongalia Counties	Outcrop	900	390	9
Sewickley bed, Monongalia County	Outcrop	580	230	7
Pocahontas 3, McDowell County	Outcrop	1000	390	5

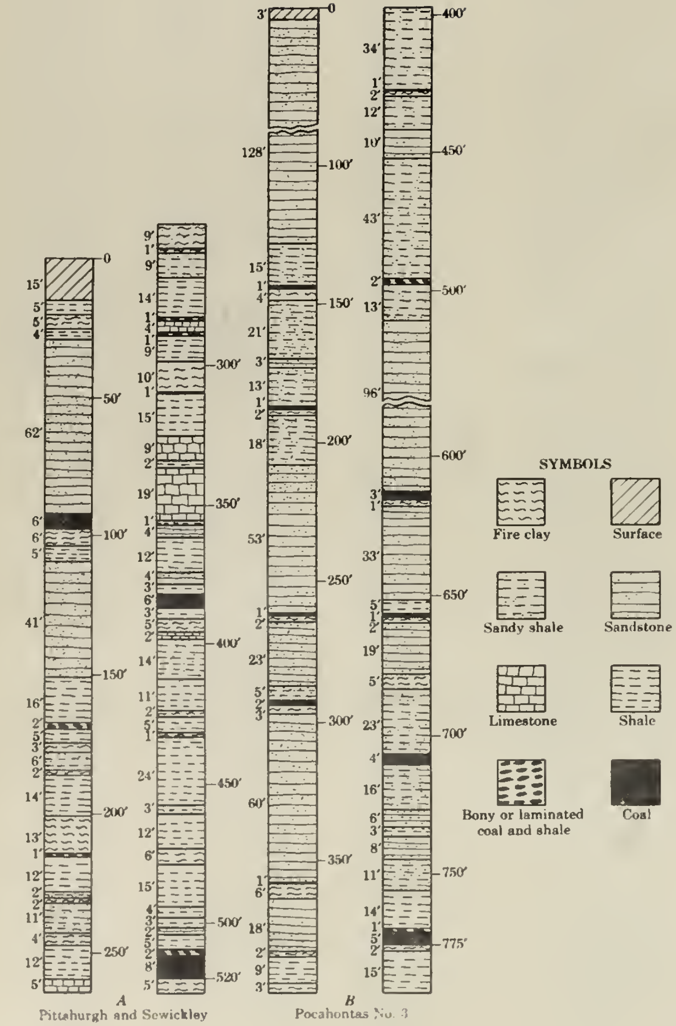

Figure 1 -Geologic section through strata overlying the Pittsburgh, Sewickley, and Pocahontas No.3 coal beds.

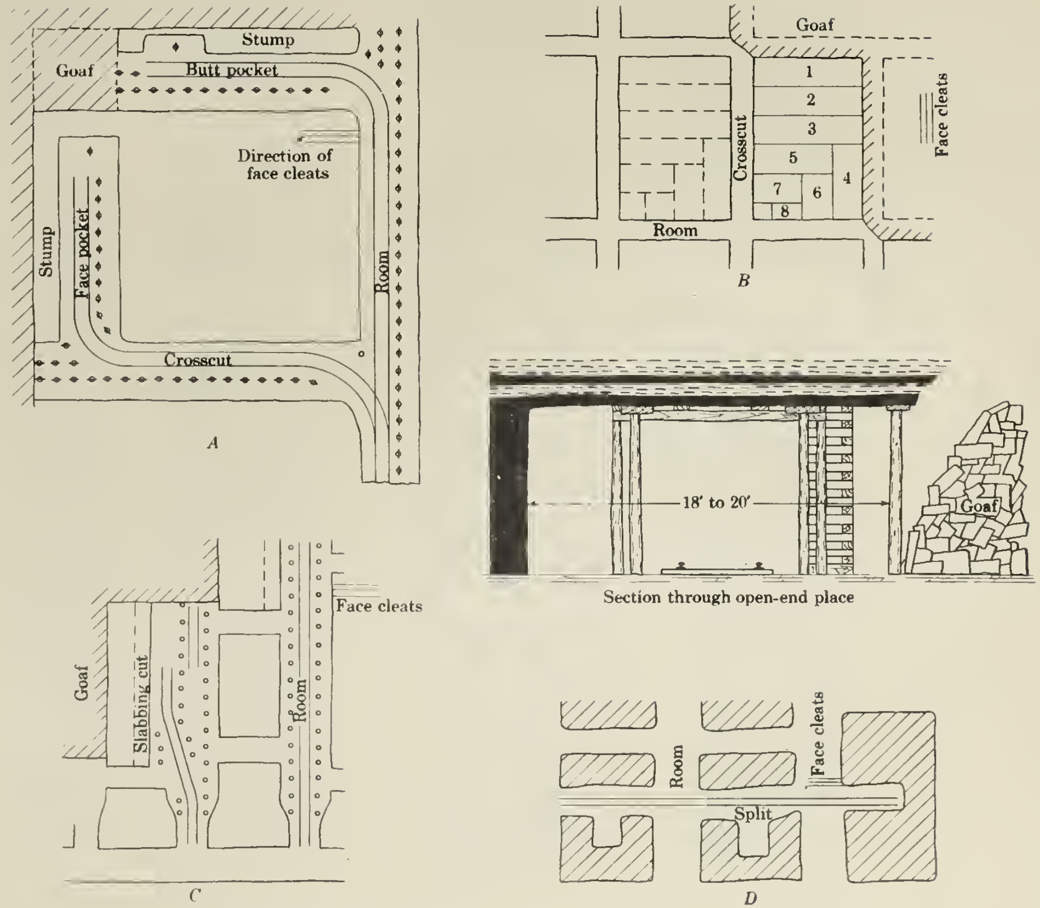

Figure 2 -General methods of extracting room pillars. A-Pocket and stump. B-Sequence of operations in open-end pillar mining. C-Slabbing. D-Splitting.

Nature of Roof

The roof over the Pittsburgh and Sewickley beds is comprised largely of sandstone, limestone, shale, and clay. The sandstone strata vary from flaggy to massive structure and may occur in beds over 60 feet thick. Limestone beds may attain a thickness as great as the sandstone and be exceptionally strong, or they may be mixed with calcareous clays and have little strength. The shales vary widely in composition and strength.

In McDowell County, W. Va., the cover over the Pocahontas No. 3 coal is composed almost entirely of sandstones and sandy shales. Beds of massive sandstone over 100 feet thick are found. No calcareous rocks are found in the New River and Pocahontas groups of the Pottsville series, which formations are exposed above the Pocahontas coal in the mines studied. Typical stratigraphic sections of the roof are shown in Figure 1.

Coal Beds

The Pittsburgh coal varies from 52 to 108 inches in thickness and is characterized by its bright appearance, hard texture, and blocky fractures, the latter resulting from the pronounced face and butt cleats in the coal. It is of bituminous rank in the region discussed and varies from a steam and domestic quality fuel to an excellent coking coal for metallurgical uses.

The Sewickley coal lies approximately 100 feet above the Pittsburgh coal in the mines studied and ranges from 45 to 72 inches in thickness. In appearance the coal is similar to the Pittsburgh and is widely used for steam and domestic purposes.

The Pocahontas No. 3 coal, which is 55 to 98 inches thick in the mines studied, is of semibituminous rank, extremely friable, and has a columnar fracture; it is not characterized by well-defined cleats, as are the former coals. This coal is widely used in making metallurgical coke and for raising steam where low ash and high calorific values are required.

Floor

The material underlying the Pittsburgh coal varies from a hard limestone several feet thick to a fire clay that becomes soft and plastic when exposed to moisture.

The Sewickley coal rests on a stratum of hard fire clay, about 6 inches thick, below which is a sandy shale; in some instances the clay is replaced by a hard, sandy shale. The clay has a tendency to expand as the coal is removed in advance work, and in a few of the mines the bottom heaves badly in places that have stood for long periods.

Underlying the Pocahontas coal is a hard fire clay about 24 inches thick, sometimes replaced by a hard gray shale. The floor does not heave readily nor is it affected appreciably by water.

Mining Methods

A detailed description of the methods of mining used would require separate treatment; however, any discussion of pillar recovery must of necessity be prefaced with a brief review of the methods used in developing the properties.

Some form of the room-and-pillar system was used in all of the mines studied. The types ranged from wide rooms and narrow pillars not recovered to the block system with narrow rooms and wide pillars which are systematically recovered. Rooms varied in width from 10 to 24 feet and room pillars from 9 to 11 feet. In general, where the rooms are wide the pillars are narrow; in a mine leaving 9-foot pillars the rooms are 21 feet wide.

The mines opened in the Pittsburgh coal and most of those in the Sewickley bed have been developed with the rooms driven on the face cleats in the coal - which means, that the face of the room is parallel to the face cleats in the coal. The room entries are driven parallel to the face cleats or on the butts and are turned off face-entry systems 600 to 2,400 feet apart. These face entries, together with main butt-entry systems, driven parallel to the room entries and connecting the face entries at intervals of 1,200 to 4,000 feet, divide the mine into panels. Headings in each entry system are driven on centers of 30 feet to 125 feet and 9 to 18 feet wide. Several of the mines using the room-and-pillar block system drive headings, rooms, and crosscuts the same width and on the same centers, generally 80 to 100 feet.

The cleat in the Pocahontas No. 3 coal is poorly defined and is not considered when developing a property; the dip and form of the property are controlling factors. Rooms in the mines visited were 16 to 25 feet wide, leaving pillars 40 to 50 feet wide. Headings generally are driven 12 to 18 feet wide, but in a few instances haulage headings have been driven 20 feet wide and air courses as much as 24 feet wide. Headings are on centers of 60 to 70 feet. The mines are generally laid out in panels similar to the mines in the Pittsburgh region unless the coal area is broken as the result of outcrops, in which case the practice has often been to make several entrances along the outcrop connected with an outside tramway.

The coal mined in development and a large portion of the pillar coal is mined with coal-cutting machines. In only one large mine was all of the coal hand-mined. Most of the coal is undercut on the floor with shortwall-type machines; in the other mines, some form of track-mounted mining machine is used, cutting either at the top or on the bottom. A few of the track-mounted machines employed are equipped for shearing, but this feature is not always used.

Mechanical loading is used to a small extent, but in most instances is still in the experimental stage. Although this method generally has proved sufficiently successful, even in pillar recovery, to warrant serious consideration, its economic use often depends on a shortage of mine labor, and the decreased demand for coal during the past few years has not been conducive to widespread adoption of mechanical loading.

GENERAL CONSIDERATIONS IN RECOVERY OF PILLARS

Essentials to successful pillar mining are the removal of all standing coal along the goaf that might delay or prevent free roof subsidence and complete control of the roof weight along the break lines. With these facts in mind, there are several factors that should be considered carefully by the management of any coal mine before attempting to recover pillars and, if possible, before extensive development is done. The factors controlling pillar recovery may be grouped under three heads: (1) natural conditions, (2) economic factors, and (3) safety considerations.

Natural Conditions

The principal natural or physical conditions that affect coal mining are the cover overlying the coal, the coal bed worked, and the strata underlying the coal that may be affected by or itself affect the removal of the coal. The overlying strata or roof is considered in two parts: the immediate roof, or that part first above the coal which is not normally self-supporting except over limited areas, and the main roof which is comprised of that part of the overburden above the immediate roof and is considered self-supporting over the working places. As behavior of these in mining is different, it is necessary to discuss each individually.

Main Roof.- The main roof may affect mine development if the cover is heavy and a strong, massive rock stratum lies above a weaker stratum near the coal. In this event the massive formation acts downward more or less equally over a large area and if the pressure is sufficient will cause the weaker material to flow into the passageways opened in the coal bed. This condition generally exists only in entries driven into virgin coal and rarely, if ever, affects pillar recovery. The factors most to be considered in planning pillar extraction are the composition, structure, strength of the strata individually and collectively, and the weight of the overburden.

Coal deposits were laid down during periods of sedimentation and as a result the strata adjacent to the coal beds are sedimentary in origin except in some districts where local igneous intrusions have occurred. The rock strata most commonly associated with bituminous coal in Pennsylvania and West Virginia are sandstone, limestone, shale, and clay.

The sandstones, if of massive formation, may be difficult to break and have a tendency to bend so as to cause a squeeze, especially if any coal is left standing in the goaf, but those beds having weak structure break readily and are seldom difficult to control.

The calcareous muds which formed limestones were often deposited at such depths that they were free from superficial currents [illegible] and the interruptions that accompanied the deposition of shales and sandstones in the shallower water; as a result, the pure limestones are relatively thick-bedded and strong. Earthy limestones, which are frequently encountered in the areas studied, may be little stronger than a clay deposit or have a strength approaching that of a pure thick-bedded limestone. Thick strata of limestone may therefore be difficult to break and, if obstructions to complete subsidence are left in the goaf, the limestone may be supported sufficiently before the elastic limit is reached to prevent fracture and result in a squeeze. The muds forming shale were often deposited in shallow water within reach of swift or variable currents which, with frequent changes in the composition of the muds, produced marked stratification. As a result of stratification and jointing, the shales are generally broken without difficulty.

Clays encountered in the roof behave much like weak shales until exposed to moisture, after which they disintegrate rapidly; ordinarily they have little effect on mining unless they occur in the immediate roof or in the floor.

Immediate Roof.- The immediate roof is a factor in mining that must not be overlooked in both development work and pillar recovery. In general, the immediate roof strata must be supported until the pillars are withdrawn or taken down as the working places advance into solid coal as is true often with draw slate. The immediate roof generally consists of shales, clays, and coal and is seldom more than 10 feet thick. Clay, such as that forming the draw slate over the Pittsburgh coal, as a rule must be taken down as exposed unless the coal bed is thick enough to permit leaving a few inches of head coal to protect the clay from the mine atmosphere. If a stratum of coal several inches thick occurs in the immediate roof, it will often prevent weathering of weaker strata above, although its actual strength is not great.

In pillar work the behavior of the immediate roof depends, to a large degree, on the action of the main roof. If the main roof breaks close to the rib line and does not cause appreciable bending over the pillars, little trouble need be expected from the immediate roof. Another important factor is the time interval between exposing the roof in rooms and the subsequent pillar extraction; the longer this interval the more weakened becomes the immediate roof, with increased hazard and expense when the pillars are ultimately extracted.

Coal bed.- The thickness of the coal bed, the structure of the coal, and its strength are important factors in the recovery of pillars. The thickness of the coal is of importance in determining whether or not pillars can be recovered. If the lowest member or members of the main roof are of massive formation and bend appreciably before rupture, it is possible in mining thin beds for the roof to rest on the floor before the elastic limit is reached, resulting in an unavoidable squeeze. Under such conditions, some form of longwall mining might be advisable.

Most bituminous and some semibituminous coals have vertical joints or cleavage planes along which the coal, in developed areas, tends to open when under heavy roof loads. Coals with poorly defined joints generally have greater rigidity than coals with well-defined cleats. The effect of structure on the strength of coal in pillars is discussed more fully in another part of this paper.

The strength of the coal in pillars is of great importance if the pillars are to be mined, and an early attempt should be made by the management of each mine to determine the amount of coal that must be left in pillars along the rib line in order to break the roof and prevent its weight from riding over the pillars.

Floor.- The floor may or may not be a factor of consequence in pillar recovery. If the floor heaves to such an extent that grading must be done to maintain the track in serviceable condition, the removal of pillars may be delayed at some points along the pillar line, introducing irregularities in the breakline. Occasionally, a floor material that flows readily under the roof weight may cause narrow pillar stumps to overturn partly, making extraction difficult and hazardous.

Economic Factors

Pillar extraction is not always economically practicable. Some of the more important factors to be considered are the value of the coal in place as compared with the value of the surface which might be damaged by subsidence, the value realized from the coal against the cost of recovery, and the possibility of water's entering the mine through surface breaks or from water-bearing strata overlying the coal. These factors should be considered carefully before extensive development is done, as changes made after a property has been largely developed are seldom entirely satisfactory. An additional factor that must be considered in mines where a large number of rooms have already been developed is the cost of reconditioning the places for pillar recovery. This will vary depending on the behavior of the roof and floor and the length of time since the rooms were developed. The principal items of expense are relaying track, retimbering, and removing falls.

In a region where the surface is valuable and subsidence would cause damage greater than the value of the coal would justify, pillar mining could not be considered. Where the coal is extremely valuable, as in the Pennsylvania anthracite region, the open area resulting from the removal of coal may be filled with sand or other waste material to prevent destructive subsidence and allow maximum recovery.

If a heavy draw slate overlies the coal and must be taken down in all working places, the cost of handling this rock in pillar work may be as much as the profit realized from the sale of the coal. In this case, however, pillars may possibly be mined profitably during periods of coal shortage, although intermittent pillar mining is not generally successful.

Where the coal is under cover of less than 100 feet, there is always danger of large inflows of surface water during rainy seasons if the roof is broken. This has caused the abandonment of pillar recovery in some mines where other conditions were favorable to pillar mining.

In certain localities large quantities of water are contained in the roof strata, which, if broken, would allow much of this water to find its way into the mine. If a thick bed of clay lies between the coal bed and water-bearing formation, the water may be sealed off by the flow of clay into the break, but, as additional roof breaks are made, water will again enter the mine, making a continual problem. In farming regions, to break a water-bearing formation often drains the wells in the district and brings damage suits or injunctions against the mining company.

Where rooms are driven as the entries are advanced and pillar extraction is delayed, the cost of recovering the pillars increases until in some instances the cost makes recovery prohibitive. This is especially true if heavy falls have occurred in the rooms and the pillars are too narrow to admit of driving a split parallel to the room to reach the back of the pillar.

Safety Considerations

The relative safety of a method of mining is sometimes overlooked until a series of accidents has forcibly focused attention on the hazard. This is particularly true in some forms of pillar recovery where the workmen are required to work under roof heavily stressed or where large open areas exist along the break line.

The greatest hazards to the workmen in pillar mining are falls of roof and coal from the sides. These may be minimized, first, by choosing a method of pillar recovery that most nearly controls the roof stresses incident to breaking the roof and prevents the carrying of excessive weight by the pillars; second, by an effective system of timbering that affords ample protection from falls of the immediate roof and spalling coal from the sides; and third, ample supervision by competent officials, without which no system of mining or timbering will be effective.

METHODS OF EXTRACTING PILLARS

The common methods of extracting pillars in the districts studied are (1) the pocket-and-stump; (2) open end; (3) slabbing; and (4) splitting. The pocket-and-stump method is the most generally used of the four and the open-end method is second in favor. The slabbing and splitting methods are seldom used in systematic recovery unless in conjunction with some other method.

The pocket-and-stump and open-end methods are similar in general principle, the greatest difference being in the method of supporting the roof along the rib line. In the former method a narrow pillar of coal known as

the stump is left along the goaf to support the roof while driving the pocket (see fig. 2A); with the open-end method, however, the pocket is open on the goaf side and the roof is supported on timber as in Figure 2 B. When the pocket-and-stump method is used, it will be noted that the workman's retreat is always protected by coal even during removal of the stump, but in the open-end method the roof on the goaf side is carried on timber which must be removed after the coal has been mined across the end. A choice between these two methods depends, to a certain extent, on the behavior of the roof and character of the coal, neither of which has been studied sufficiently for the formulation of rigid rules; however, such observations as have been made will be discussed under the behavior of pillars along the rib line.

An example of the slabbing method for the recovery of pillars is shown in Figure 2 C. By this method successive slabs are cut from one rib after a room is finished, until as much of the pillar is removed as can be recovered safely. The disadvantage of this system is that the open area of the working place is always increasing and that the loaders are always working away from the solid pillars toward the goaf.

Figure 2 D shows a method of recovering pillar coal that is now seldom found, although its use was common in the early periods of mining. By this method a room is driven through the pillars, splitting them into smaller blocks; from this secondary room, pockets are turned at right angles and driven into the small pillars to recover such coal as may be mined safely. This method amounts to little more than gouging the pillars, and as the recovery is seldom high, the method is wasteful and generally results in a squeeze.

DESCRIPTION OF RIB LINES

In pillar recovery it is essential to relieve the pillars being mined of all possible weight in excess of that resulting from the cover directly over them. The ideal method would be to cause the roof to shear vertically along the pillars as the coal is removed, but as this would be extremely difficult to accomplish in practice, the solution lies in extracting the pillars in a manner that will produce breaks most closely approaching the ideal and at the same time will produce the least stress in the roof over the working places.

In advance mining and during the early stages of pillar extraction the roof behaves like successive beams, but as the unsupported span becomes greater the lower members of the roof break and fall. The removal of additional pillars again increases the open area and the lower strata of the roof are stressed like cantilever beams and will fall as the weight of the overhanging strata becomes sufficient to break the rock. Similarly, as the span is increased, the higher strata fall until the surface is reached and the entire roof may be considered a series of cantilevers. In a cantilever the bending moments and shear stresses are maximum at the point of support and, if these stresses are to be utilized fully to break the roof, the points of support

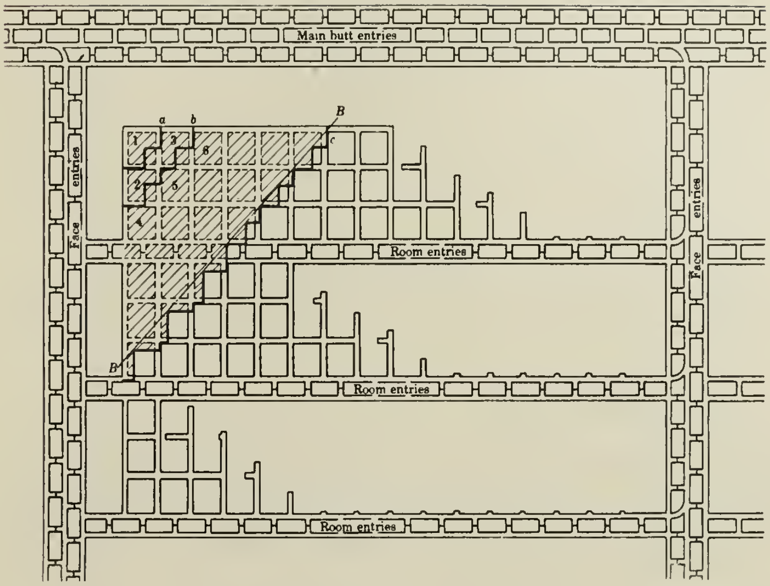

Figure 3.—Development of a single rib line.

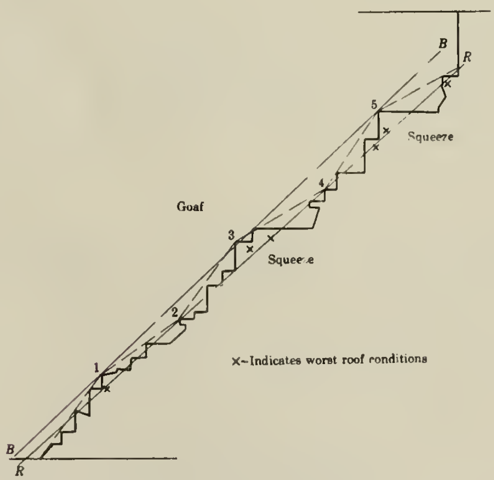

Figure 4 —Trace of irregular rib line, showing effect on roof over pillars.

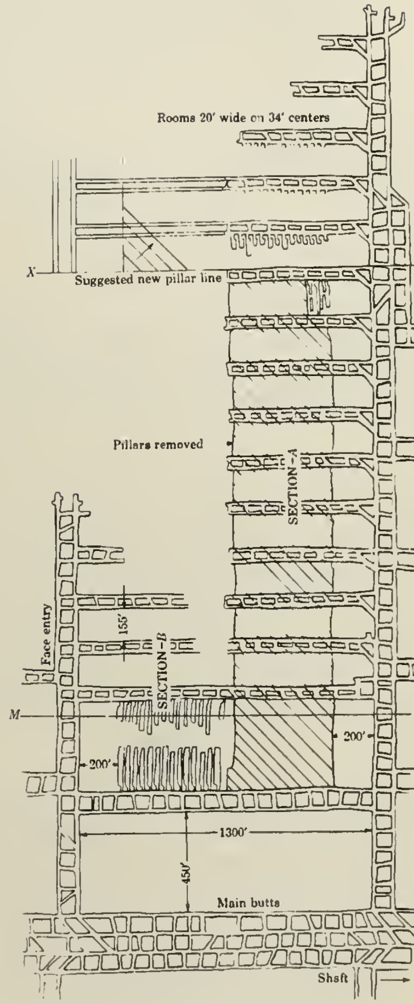

Figure 5.—Mining pillars in narrow panels with suggested method for improvement.

should lie along a straight line. As a result, pillars are generally extracted along continuous lines of varying lengths, the traces of which resemble a series of steps as shown in Figure 3. In the strictest sense, such a rib line could hardly be considered a straight line; however, the actual trace of the line is not as important as the points supporting the overhanging roof. Along such a line the roof strata, except the weaker members of the immediate roof, are supported as beams across the goaf area between the pillars; but the main roof overhanging the goaf as a cantilever is most heavily stressed at the points of support or along the straight line B-B through the corners of the pillars extending into the goaf.

The more common forms of rib lines are the single line, the compound line, the diverging V-line, and the converging V-line; of these, the single line is the most common and represents the rib line in its simplest form.

The Single Line

A single line generally is commenced by removing a pillar at one corner of a panel, like pillar 1 in Figure 3. When the extracted area reaches line a, mining is commenced in pillars 2 and 3, to be followed by the extraction of pillars 4, 5, and 6 when the goaf area extends to b.

After establishing a rib line, it is essential that the break line be kept straight. By this it is not meant that a corner of each pillar must at all times coincide with the break line, for such a condition would be difficult to maintain in practice, but it is necessary to have a sufficient number of the pillars to coincide with this line to support the roof and to distribute this support uniformly along the entire rib line, because if the load is supported unequally, the roof will crush the overloaded pillars and develop into a squeeze.

The effect of an irregular rib line is shown in Figure 4. This rib line is approximately 2,400 feet long and under heavy sandstone and sandy shale cover. It will be noted that at points 1, 3, and 5 the pillars coincide with line B-B, which should be the break line if the pillars are to be relieved of excessive roof weight. However, as these points are approximately 750 feet apart and the cover is 500 to 700 feet thick, the load is too great for the coal to support, causing the pillars at these points to crush and breaking the roof over the working places. If the cover were light and the rock structurally weak, a series of short break lines connecting points 1 to 5 with the ends of the rib lines might prove reasonably satisfactory, but a strong roof strata will act along a straight line over the pillars, the exact position of which will be determined by the strength of the supporting pillars and the weight of the overlying strata. It will be assumed that this line lies along R-R; then on the goaf side of this line the pillars will be crushed appreciably and the roof over working places will show signs of failure. An examination of the working places along this line disclosed that the pillars at points 1, 3, and 5 were badly crushed, rendering recovery hazardous and

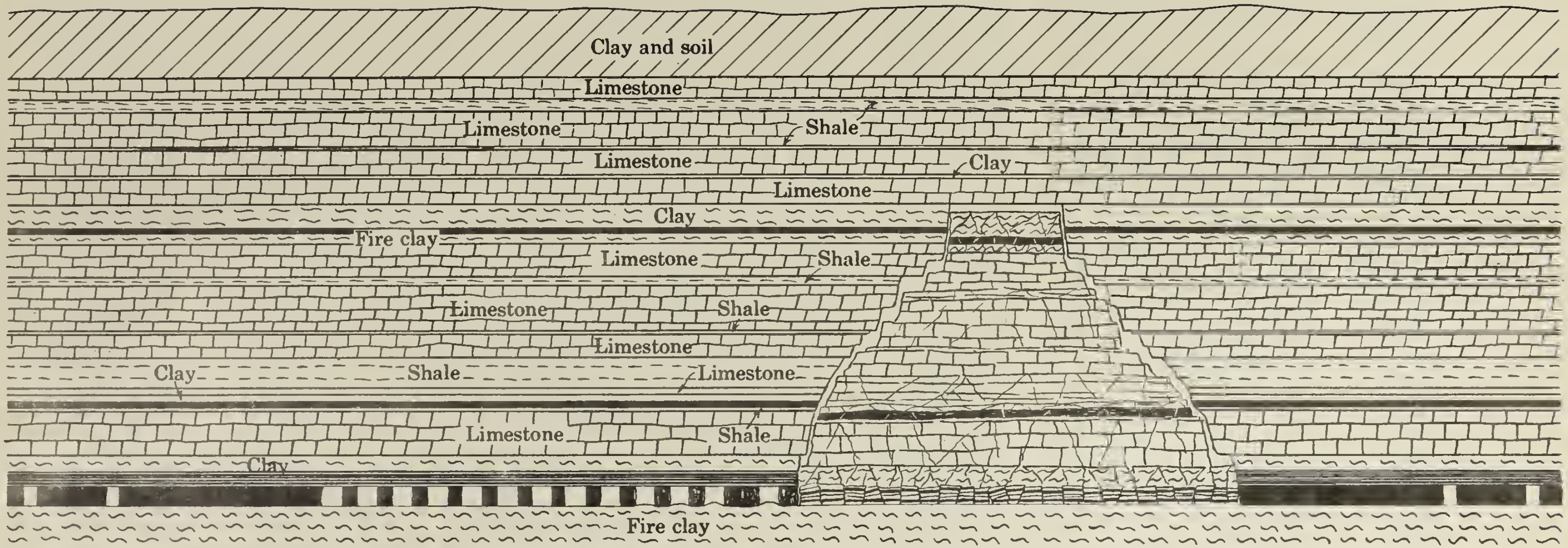

Figure 6.-Section through M-M, Figure 5, showing probable behavior of the roof.

expensive, and that the roof over working places was broken and often fell. The points where the squeeze was most apparent on the pillars and roof are designated by crosses (X) on the diagram.

Opinions differ as to the most satisfactory length for rib lines and, in the mines used as a basis for this report, rib lines less than 500 feet and up to 4,000 feet long were being maintained. The principal factors determining the length of rib lines are the strength and thickness of the roof, control of ventilation, and the production requirements. Only the first of these has any direct bearing on the control of the roof; the second is purely a safety measure and depends on the size of panel that can be ventilated safely by a separate split of air; the third factor, depending on production requirements, becomes important if the output demands fluctuate widely, in which case short lines may be more satisfactory.

Careful consideration should be given in planning pillar extraction to provide rib lines of sufficient length to prevent strong roof strata from spanning the open area across the panel or section from which pillars are being extracted. An instance illustrative of this was found in a mine operating in the Pittsburgh bed, a section of which is reproduced in Figure 5.

The main roof over this mine consists of limestone, shale, and clay; the limestone comprises 60 per cent of the cover, some beds of which are difficult to break. Referring to Figure 5, it will be noted that the total distance between face entries is 1,300 feet, of which 400 feet is reserved in barrier pillars 200 feet wide along each face entry. This leaves 900 feet in which to drive rooms and extract pillars; however, instead of driving rooms entirely across this distance as a single panel, the policy has been to turn rooms and recover pillars in only half of the panel, or across section A, having a span of only 450 feet. The lower roof strata breaks readily and falls are made as the pillars are removed, but no surface breaks have been found nor other evidence that would indicate subsidence of the main roof over this section. It is therefore evident that some stratum or group of strata in the main roof can not be broken across a span of only 450 feet. This conclusion was further substantiated when a squeeze developed over section B as rooms were turned near section A. Had the roof broken to the surface over section A, the pillars in section B would have been supporting only the roof over that section, and as the pillars were extracted, the roof would have continued to break and relieve the remaining pillars. A section through the roof at M-M showing the possible behavior of the roof is given in Figure 6. It is believed that subsidence is arrested at the limestone, a short distance above the Sewickley bed. This limestone is exceptionally strong and frequently occurs as a single bed 30 to 40 feet thick.

Assuming that the roof over section A behaves as shown, the unbroken strata will be supported equally by the coal on each side of the mined area. As the rooms are driven in section B, approximately 60 per cent of the support is removed along one side of the goaf and the roof commences to bend

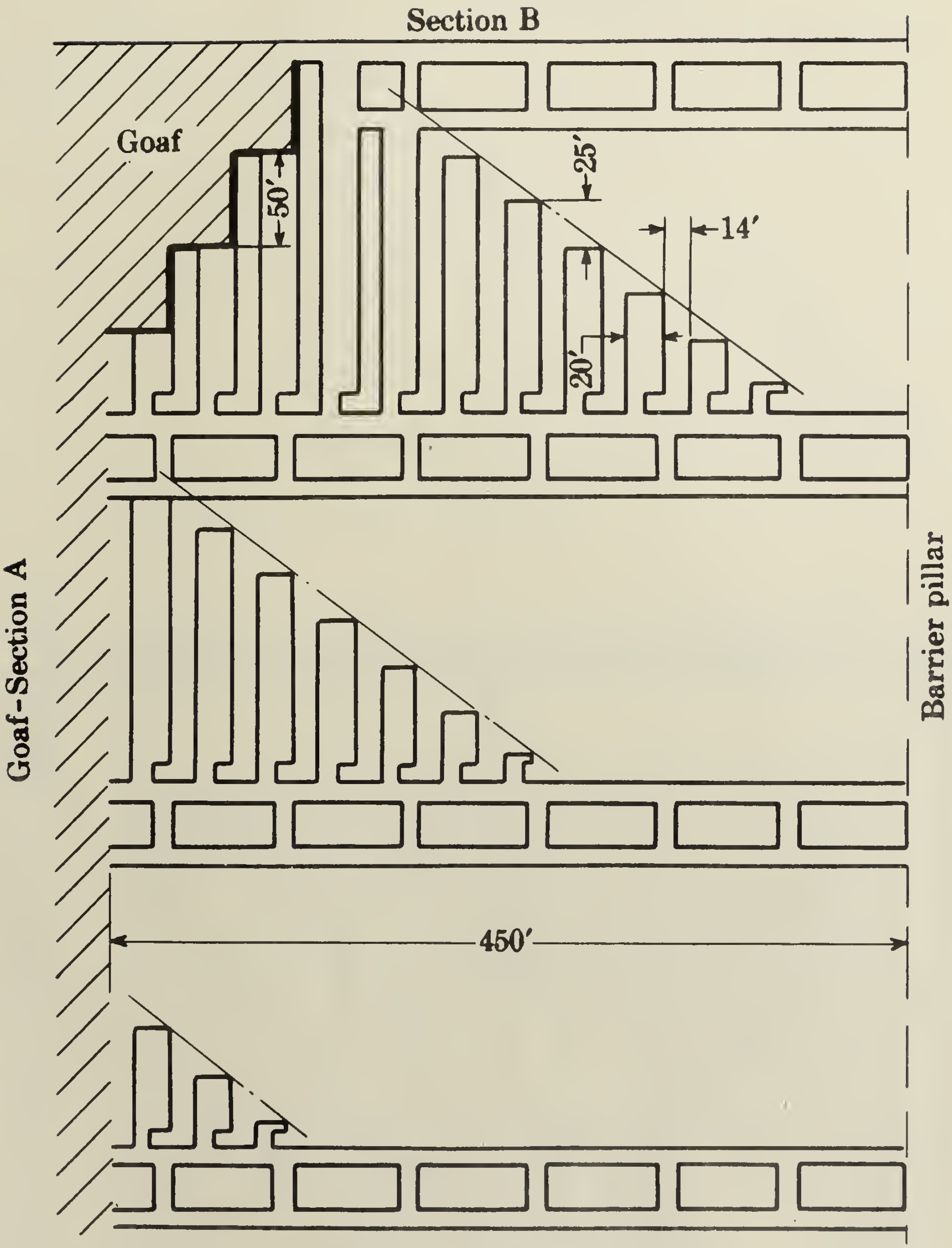

Figure 7 - Proposed method of mining section B.

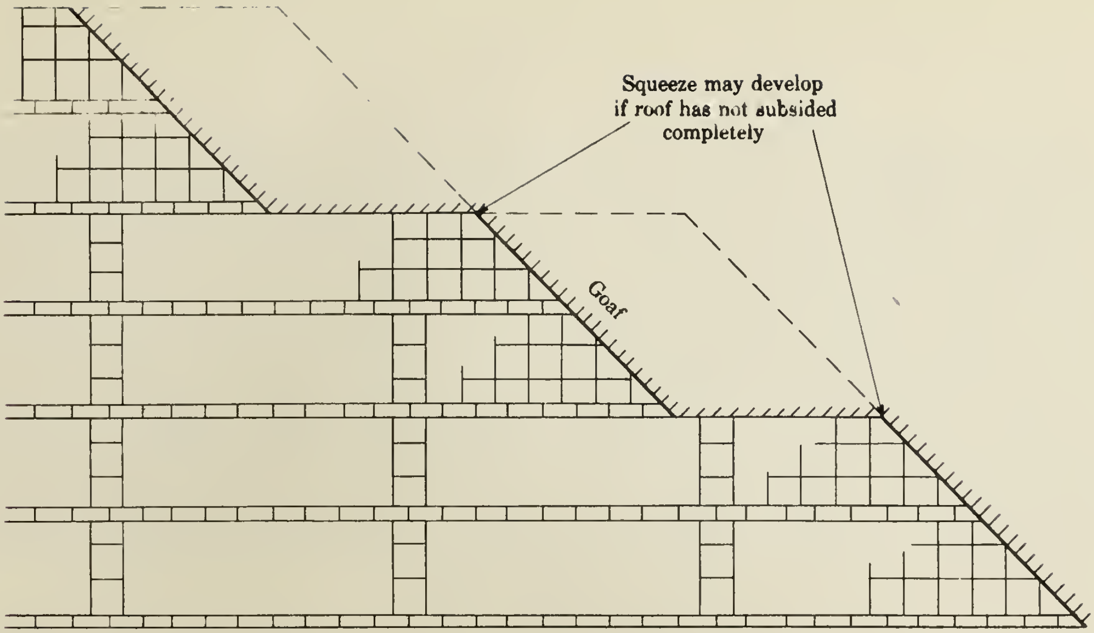

Figure 8 - General form of compound rib line.

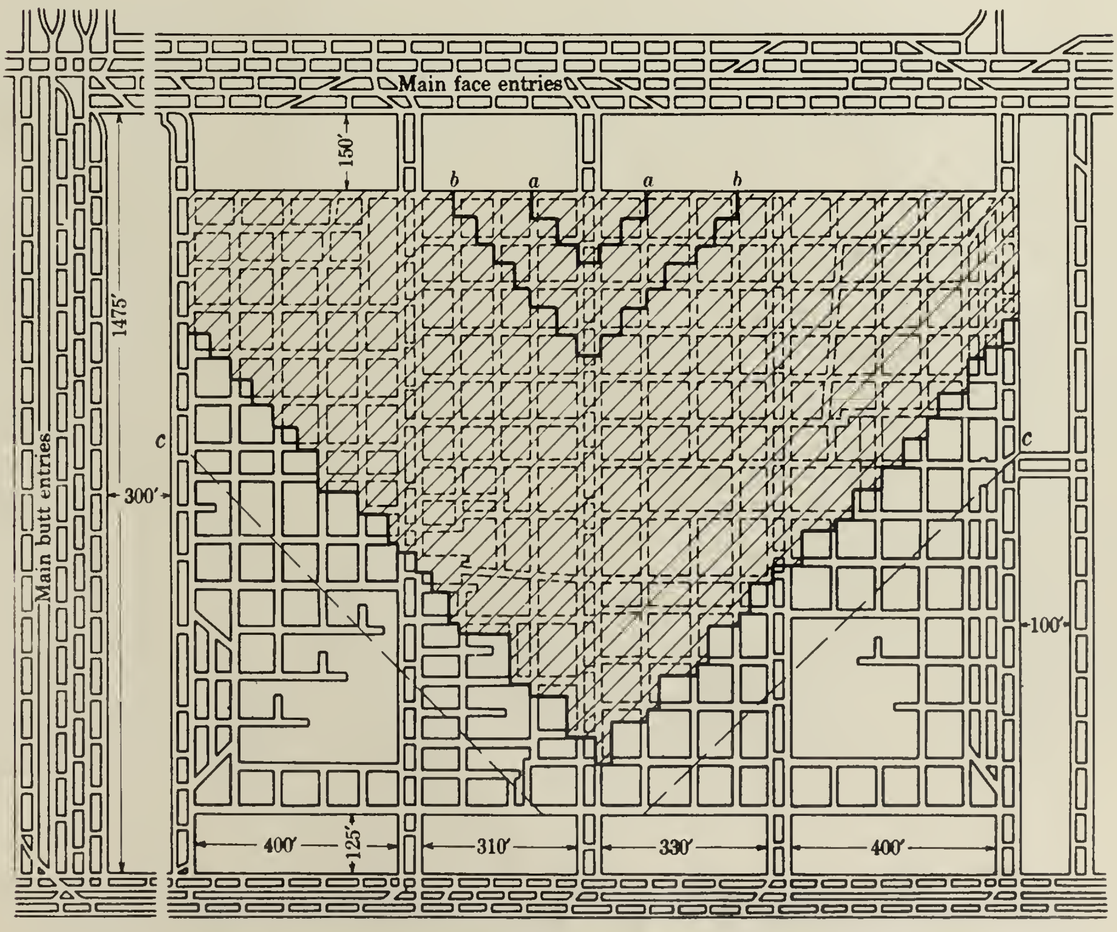

Figure 9 - Diverging V-lines.

and crush the pillars. The squeeze may be arrested by leaving a barrier pillar between sections A and B strong enough to hold the roof, but this would necessitate the loss of coal left in the pillar and might cause further loss due to falls of the immediate roof. Instead of merely arresting the squeeze and getting such coal as might be recovered safety after leaving a barrier pillar, a method should be adopted that would admit of the recovery of all coal and also break the roof, relieving excess weight from the pillars. A plan such as that illustrated in Figure 7 should prove satisfactory for this purpose, although certain modifications in the width of pillars might be necessary; however, this can only be determined by trial. After this pillar line is established, pillar extraction should be stopped in section A until the pillars in section B have reached the line X-X in Figure 5, when a new rib line should be started as indicated, continuing across the entire panel.

The Compound Line

The compound line may best be described as a series of two or more parallel single lines, separated by offsets along which the roof may be broken. A typical line of this type is shown in Figure 8. Such lines are frequently developed from a long single line when production must be curtailed, and it is considered advisable to stop part of the rib line and work the remainder more or less regularly rather than attempt to maintain the entire line during intermittent operation. Under such a plan as much of the upper portion of the line as can be worked is continued, while the lower portion is stopped temporarily. When increased production is again required, the lower portion is started, but an offset now exists in the break line which may be allowed to remain, or, if not too great, may be removed by speeding up the recovery in the lower portion. When mining under a strong roof, a squeeze sometimes develops at the offset if the lower line is started before the roof along the offset is broken. If it is planned to maintain the broken line the distance along the offset should be sufficient to allow the roof along the offset to subside and reach a state approaching equilibrium where it joins the goaf of the line following. This distance will depend on the time required for subsidence, and the rate at which the rib lines advance. If the cover is not more than 200 feet and no strata difficult to break are encountered, subsidence may be practically complete in one year; but as a rule in the districts discussed in this paper, from 2 to 3 years are required for subsidence before pillars can be mined satisfactorily against a goaf; therefore the length of offset should be two or three times the distance that the rib lines retreat annually. Compound lines are advantageous when production must be curtailed and are less difficult to maintain in perfect alinement when two or more rib bosses are required along a single line, but there is always a possibility of squeeze over the pillars adjoining the goaf of the preceding line if subsidence is still active.

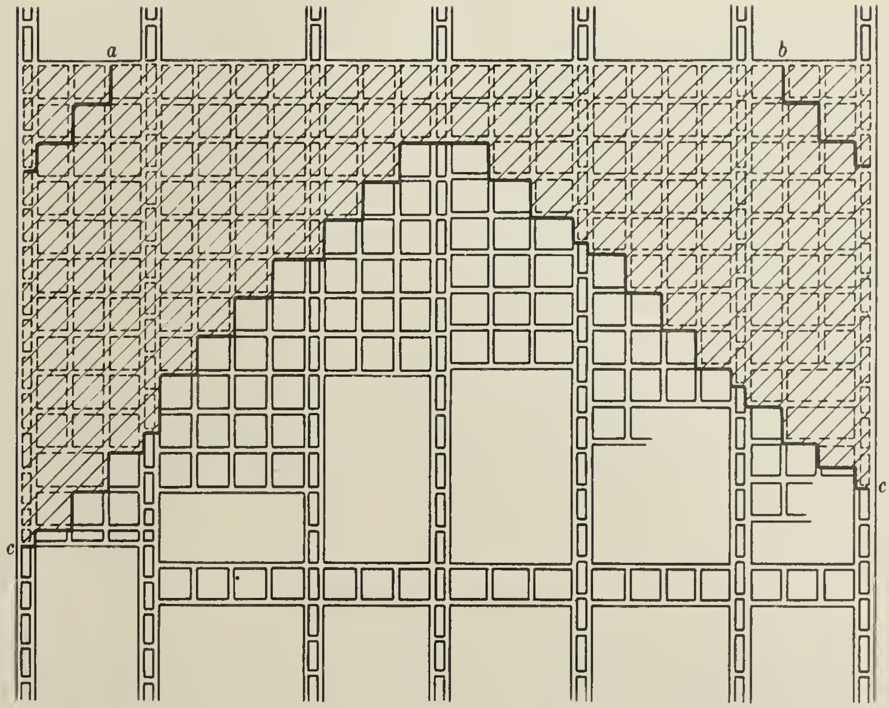

Figure 10. –Converging V-lines.

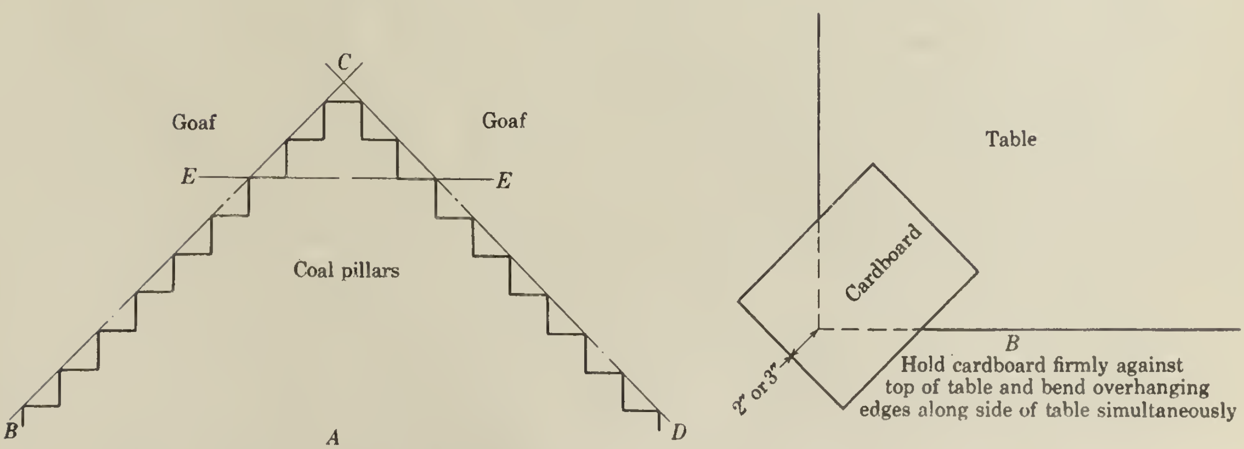

Figure 11.–Roof stresses developed along converging rib lines.
A, Trace of converging-type rib lines showing break line;
B, Experiment illustrating behavior of the roof over converging V-lines.

Diverging V-lines

Diverging lines consist of two single lines that intersect to form an angle of less than 180°, the lines taking the general form of the letter V with the goaf lying within the angle, as in Figure 9. This type of line is used frequently to obtain a large production quickly and is started by removing the pillars at the midpoint along one side of a panel until a triangular goaf is formed with its base adjoining the solid coal and the other sides forming the rib lines, as in Figure 9, lines a and b, showing successive stages of development. It will be noted, therefore, that the pillars may be extracted from a given panel in much less time by using this system than by using a single line. This system will generally prove satisfactory where the roof is easily broken, but if massive strata, self-supporting across wide spans, are encountered in the roof and the pillars are being extracted slowly, some weight may be encountered at the vertex, although it is not likely that a serious squeeze would develop with this method. When the vertex reaches the boundary of the panel, the goaf distance across the point increases until two single lines are formed.

Converging V-lines

Converging pillar lines are formed by reversing the procedure used in developing the diverging lines. As shown in Figure 10, two rib lines are commenced at the extremities of one side of a panel, as represented by a and b which move toward each other and finally intersect to form the V-shaped line c, with the pillars to be extracted lying within the angle. The two break lines common to all V-shaped rib lines lie along the outer edges of the V and intersect at a point over the goaf beyond the last pillar. Figure 11 A shows the trace of a converging rib line where B and D represent the break lines theoretically intersecting at C; however, in practice the roof, unless structurally weak, seldom breaks in this manner at the vertex. The behavior of the roof over the intersection can be illustrated by placing a piece of cardboard, 8 to 10 inches square, over one corner of a table and allowing about one third to extend over the edge, as in Figure 11, B; then, while pressing the cardboard firmly against the top of the table, bend the projecting cardboard along both edges of the table simultaneously. As this is done it will be observed that the cardboard bends readily near the extremities but, as the corner is approached, it bends less freely until the conflicting stresses from the two sides become so great that the normal bending action ceases. This experiment demonstrates the reactions that occur in any strong formation in the roof, and in a roof comprised of several such strata the stresses probably would be even more complex. As a rule these conflicting stresses prevent clean breaks as the coal is removed and increase the load carried by the strata; as a result, the pillars are crushed and unless the cover is light the weight develops a secondary break line across the vertex, as shown by E-E. The position of this line depends on the roof weight at the vertex and the strength of the pillars, and generally lies approximately 100 feet back of the vertex; but the stressed zone, as indicated by crushed pillars and broken immediate roof, may extend back over 300 feet from the point.

Conflicting Break Lines

In mines where several rib lines are maintained there is a possibility that the stresses produced along one break line will meet those set up by another line, which may, if the stresses are of sufficient intensity, cause breaks in the immediate roof at points several hundred feet from either pillar line. In one mine the roof over developed rooms near one boundary line was broken as the result of stresses set up between a rib line in that mine and one in an adjoining mine. The two break lines were parallel but traveling in opposite directions. The roof over the rooms commenced to break when the opposing rib lines were about 900 feet apart, but at the time of visiting the mines the interval was only 200 feet, and the roof, a strong shale, was broken to a height of about 3 feet.

In another mine heavy falls were occurring over a distance of about 400 feet on the four headings of a main entry system which had been driven for several years. An examination of the mine maps showed that four rib lines, two on each side of the entry, were in such position that their break lines might be extended to form an X with the intersection over these headings, which apparently accounted for the sudden failure of the roof.

In one district studied, rooms were turned off both headings of each room-entry system and the pillars were recovered by the method known as the half advance and half retreat system. With this system, the rooms on one heading are driven as the entry advances and the pillars are extracted as the rooms are finished. When the entry is finished the rooms on the other heading are started at the inby end and are followed by pillar extraction. If all the pillars on the advancing side have not been extracted before the retreating line is started, conflicting stresses will be set up between the lines, often causing the loss of coal. This method of mining is not recommended, especially where the cover is heavy.

ANGLE OF BREAK LINE

All of the break lines described are at an angle with the room faces. The primary cause for this angle is the stepping of the rib lines to break the roof over the goaf instead of over the pillars, thereby relieving the roof over the pillars of undue stresses. The angles made by the break line and room faces vary from 20° to 60°; however, in 75 per cent of the mines the break lines were maintained on an angle of approximately 45° to the room faces.

The angle of the break line is a more important factor in controlling roof breaks than is generally realized and may be a determining factor in the success or failure of pillar recovery. Most of the roof strata contain two sets of joints perpendicular to the bedding planes, one set parallel to the strike of the strata and the other parallel to the dip. As the roof strata are largely concealed from direct observation except occasionally along outcrops or along break lines, little is known of the relationship that may exist between the cleats in the coal and joints in the roof or the effect of the

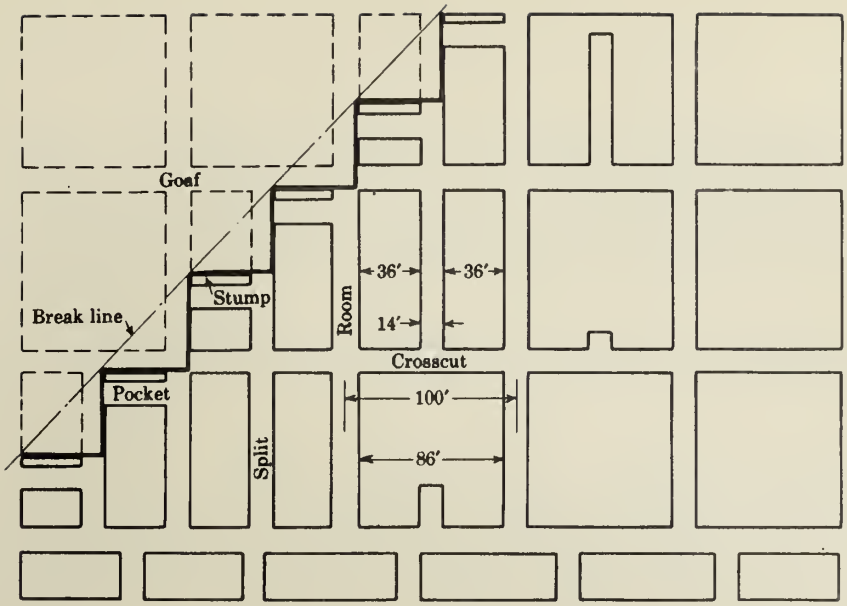

Figure 12.-Splitting room pillars to shorten steps along the rib line.

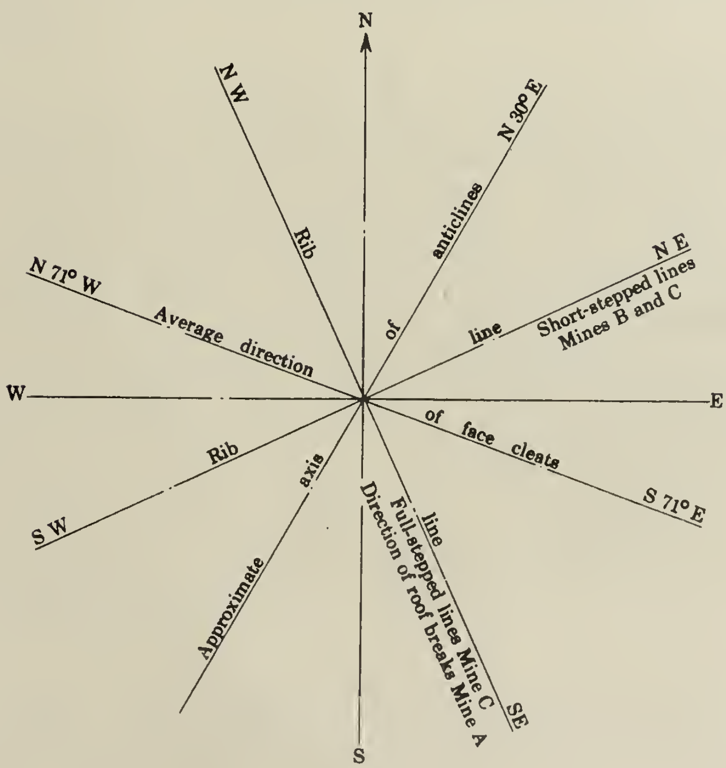

Figure 13.-Diagram showing direction of break lines with respect to cleat in coal and axis of anticlines.

roof joints in mining. In several instances, however, changes have been made in either the direction or angle of break lines which have resulted in improved roof conditions along the rib lines and indicate that pillar mining might be improved in other mines by a thorough study of the rock structure and its behavior in pillar extraction.

The following examples are illustrative of the results obtained along break lines at 45° to the room faces, but having bearings northeast-southwest and northwest-southeast.

Several rib lines were being worked in mine A, opened in the Pittsburgh coal under 450 feet of cover. About half of these lines had a general bearing northwest and southeast and the others were at 90° to these. All break lines made angles of approximately 45° with the face cleats in the coal. As the rib lines were developed it was found that the roof broke more freely along the northwest-southeast lines, and when breaks appeared on the surface their courses had approximately the same direction, even when over the break lines running northeast and southwest. To take advantage of the better roof breaks along the northwest-southeast rib lines, the management now plans to discontinue the rib lines at 90° to these as rapidly as possible, and in new development proposes to extract pillars along lines parallel to the planes of weakness in the roof.

In mine B, opened in the Pittsburgh coal under cover averaging 150 feet, the rib lines are maintained at an angle of approximately 45° to the room faces or face cleats in the coal and have a general direction northeast and southwest. Lying a few inches above the coal over a large part of the property is a bed of hard sandstone and sandy shale, 20 to 50 feet thick, having pronounced joints 15 to 25 feet apart, their course being approximately 45° to the face cleats and normal to that of the break line. These joints cause the roof to break in large blocks whose weight sometimes crushes the corners of pillars extending into the goaf. To prevent this, it was decided to strengthen the pillars along the break line rather than change its direction, as it was believed advisable to keep the break line normal to the joints rather than parallel to them. This was accomplished by splitting the pillars parallel with the rooms as shown in Figure 12, thus reducing the maximum offsets along the rib line from 100 feet to 50 feet and reducing the goaf area between steps proportionally. Since this method was adopted the corners of pillars have been crushed less and the roof appears to be less heavily stressed over the working places.

Mine C, opened in the Pittsburgh coal, has two rib lines, each under about 450 feet of cover. Both lines are over 2,000 feet long and cross the face cleats in the coal at an angle of approximately 45°; one line has a bearing roughly northeast and southwest and the other, 90° to the first, is northwest and southeast. The lowest stratum of the main roof over the mine consists of 35 to 40 feet of strong sandstone. This is separated from the coal bed over the northwest-southeast rib line by a few feet of shale and coal, but

over the other rib line the sandstone is only a few inches above the coal. As the latter rib line was developed, much difficulty was experienced in breaking the roof. After several methods of extracting the pillars had been tried it was found that best results were obtained by shortening the steps along the rib line while maintaining the 45° break line. This was accomplished as in mine B, except that the pillars were mined by the open-end system instead of by pockets and stumps. No difficulty was experienced on the northwest-southeast rib line using the full block system, although splitting the pillars was tried.

The results of these observations in the three mines are shown in Figure 13. In this diagram the average bearing of the face cleats has been used, as has also been done to obtain the bearing of the anticlinal axes. As all of the rib lines were at 45° to the face cleats, they may be represented by the two lines intersecting at a right angle as shown. It will then be observed that the rib lines in mine A, along which the roof breaks more freely, have the same bearing as the full-stepped line in mine C and that the short-stepped line in C and the short-stepped lines in mine B have the same bearing. Mines A and B are in West Virginia and mine C in Pennsylvania. The two in West Virginia are about 10 miles apart, while mine C is over 40 miles from the former.

In the mines of one company operating in the Pittsburgh bed in Pennsylvania, it has been found that better roof breaks are obtained and less difficulty is experienced with the immediate roof in working places along the rib lines by maintaining the break lines at angles of 30° to the butt cleats, where places are driven on the butts, and 70° to the butt cleats where the places are driven on the faces. Little is known of the main roof structure in these mines, and it is not known whether the joints in the main roof or those in the coal are responsible for the choice of these angles.

In several mines having a strong, tough roof it is necessary to have the greatest possible support extended to the break line to prevent the strata from bending excessively over the working places. This may be done along a 45° break line by shortening the steps as in Figure 12, or it may be effected by changing the angle of the break line as in Figure 14 in which two full-stepped rib lines, one at 45° and the other at 30°, are compared. An examination of these show that the unsupported area A between the pillars when rooms are on 100-foot centers is 5,000 square feet along the 45° break line and 2,900 square feet along the 30° line; the distance c between points of support along the breakline B-B is 141 feet for the former and 116 feet for the latter; and the distances d from S-S, the line of maximum pillar support to the break line, are 71 feet and 50 feet, respectively.

This distance d is an important factor in breaking the roof and in maintaining a sound roof over the working places along the rib line. It has been stated that with the roof behaving as a cantilever, the bending moments and shear stresses are maximum along the points of support; as the maximum pillar

support is along S-S and gradually diminishes to B-B, there will be a certain amount of bending over the pillars between S-S and B-B. Obviously, this will be greatest along B-B and will decrease to practically zero along S-S. It is therefore evident that as the distance d becomes shorter, the roof stresses will become more concentrated, facilitating the fracture of the roof and reducing the area over which the immediate roof will be disturbed.

Rooms and crosscuts have been shown on 100-foot centers; however, the same ratios will exist for other centers or where the pillars have been split, the only difference being that shortening the room centers reduces the proportion of solid coal along the rib lines and is equivalent to widening the rooms without changing the centers.

BEHAVIOR OF PILLARS ALONG RIB LINES

It has been shown that a certain amount of bending occurs over the pillars adjoining the rib line. The extent to which the roof bends is controlled by the distance between the break line and the line of support and the resistance afforded by the coal pillars. The first factor may be controlled by methods already described; control of the second depends on the structure and strength of the coal.

The vertical joints or cleats commonly found in the coals of bituminous and semibituminous rank are probably the most important structural features to be considered. These joints, although sometimes more or less obscure, generally occur in two sets intersecting at approximately 90°. The longer and more pronounced of these joints are known as the face cleats, while the others, which are shorter and less defined, are known as the butt or end cleats.

As a rule, when a pillar is subjected to excessive load, cracks develop along the face cleats and pronounced spalling of the coal occurs along the rib parallel to these joints. The depth to which the pillars are affected depends on the load and the degree of development of the joints. This is particularly noticeable in pillar work where the pocket-and-stump method of extraction is used. Referring to Figure 2 A, which is representative of this method, it will be noted that in the stump along the pocket turned off the room the face cleats are parallel to the longer sides, and in the stump adjoining the pocket driven off the crosscut the face cleats are parallel to the ends. After observing the behavior of coal in both types of stumps in both the Pittsburgh and Sewickley coal beds, the writer is convinced that in two congruous rectangular pillars, one with the face cleats parallel to the longer sides and in the other parallel to the shorter sides, the former will offer less resistance to roof pressure than the latter and this difference will become greater as the length of the pillars increases with respect to the breadth.

The extent to which the arrangement of the cleats in stumps affects the main roof is not definitely known, provided that a straight break line is maintained; however, stumps in which the face cleats are parallel to the shorter sides are known frequently to break the immediate roof over pockets

driven in pillars in the Pittsburgh coal. In many mines this effect has been overcome by making such stumps narrower than those in which the face cleats are parallel to the longer sides. In most mines in the Pittsburgh coal it is possible to leave stumps 12 to 15 feet wide where the face cleats are parallel to the longer sides, while the stumps at right angles to these generally must be kept less than 6 feet wide in order not to break the immediate roof over the face pockets. In a few mines the difficulty attendant upon leaving stumps along the face pockets is eliminated by driving only butt pockets, along which it is possible to leave stumps two machine-cuts thick; however, this method is not always advisable, as it is sometimes necessary to keep the pillar blocks as nearly square as possible. One company has managed this in a most commendable manner by driving butt pockets off the rooms, leaving stumps 12 feet wide, and by driving face places on the open end, leaving a thin curtain of coal not to exceed 2 feet thick to prevent rock in the goaf from sliding into the working place. After each face place is driven across the end of a pillar, two slabbing cuts are made from the pillar, as in Figure 15. This method permits the removal of as much coal from the end of the pillar as would be taken if mined by pockets, leaving 12-foot stumps.

The poorly defined cleats in the Pocahontas No. 3 coal apparently have no effect on the method of mining; however, the friable nature and columnar structure of this coal causes it to behave along all ribs much as does the harder Pittsburgh coal parallel to the face cleats. At the vertex of converging rib lines in one mine where the coal was 7 feet thick and under 500 feet of cover, the coal along the ribs was badly crushed and spalling from the sides, but when the pockets were driven to a depth of 6 feet in the pillar no visible evidence of crushing was found. This would indicate that with the pocket-and-stump method of pillar extraction, stumps 12 to 14 feet wide might be used successfully. This assumption was borne out in three mines where 12-foot stumps were the standard. In another mine the stumps were 6 to 8 feet thick and were apparently causing no difficulty, although they were sometimes crushed more than when wider. In one mine the pillars were removed on the open end with fair results. In this mine the recovery was high, but the working places were in worse condition generally than those found in an adjoining mine having the same management but where the pocket-and-stump method was used.

MAINTAINING RIB ALINEMENT

After establishing a rib line every effort must be made to maintain a straight break line. If pillars are recovered as the rooms are finished, adequate development must be provided to preclude delays along the rib line while waiting for rooms to be finished. Conversely, if all rooms have been driven before commencing pillar recovery, every effort should be made to obviate delays due to reconditioning the rooms for pillar mining. Falls and water are causes of delay that may be avoided or mitigated by regular inspection and by keeping necessary supplies on hand. Lack of adequate supplies is a common cause of delay in many mines. Every effort should be made to anticipate the operating necessities.

Competent supervision is essential in all underground work and it becomes imperative in pillar extraction if best results are to be obtained. As an aid to the pillar bosses, guide lines parallel to the break line may be drawn on the maps furnished them. The points on the pillars where these lines intersect the ribs should then be marked with white paint and given a number or letter corresponding to that of the line on the blueprint. These lines should be spaced to represent not more than 50 feet and if possible should be spaced to represent the distance that the pillar is shortened by each successive pocket-and-stump or open-end cut. With such guides there should be no excuse for irregularities in the rib line, provided that it was not delayed by lack of development, extensive falls, or water.

Several mines have adopted a plan known as the "clean-up" system that makes possible a more uniform rate of advance or retreat. This system embodies the principle of providing each loader with only so much coal for loading as he can dispose of during the shift, along with his other duties. In one mine opened in the Pittsburgh bed each loader averaged 19.3 tons per 8-hour shift in addition to drilling three holes, setting one permanent post, and advancing his track. No draw slate is handled, as a stratum of coal is left up to support it. With such a system, all working places are cleaned up at the end of a shift and are ready to be cut on the night shift, so that the foreman is able to estimate accurately the rate of advance or retreat of each place.

COMMON CAUSES OF FAILURE IN PILLAR MINING

Pillar recovery has been unsuccessful in some districts or in individual mines, giving rise to the belief that pillar extraction is impossible in those places. This may be true in a few instances, as in thin coal beds overlain with a tough roof that bends to the floor without breaking or where fracturing the roof would cause serious inflows of water. Usually, however, disappointing results in pillar extraction may be attributed to the failure of the management to consider carefully all of the factors that affect pillar mining. In one district where several unsuccessful attempts have been made to recover pillars, the failure may be attributed to the following causes: (1) Leaving room pillars too small; (2) making rib lines too short; (3) improper choice of pillaring method; and (4) incomplete recovery.

If pillars are to be recovered, they should be at least as wide as the rooms and wider if the cover is heavy. It is interesting to note that in those districts where pillar recovery is most systematic, the ratio of the room widths to the pillars varies from 1:5 to 1:9 and averages 1:7; whereas in the districts where pillar extraction is considered impossible, the ratios are from 1:0.4 to 1:0.5.

The length of the pillar line as a factor in successfully breaking the roof has been fully discussed, but it is well to add that more pillar mining has failed because the rib lines were too short than because they were too long.

Every effort should be made to find the pillaring method most suitable to conditions in the mine. In the mines where best results were attained in recovering pillars, the pocket-and-stump or the open-end method or a combination of the two methods was used, whereas in those mines where pillaring was considered a failure, the slabbing method was used most frequently. If the rooms are driven on the face cleats and the pillars are too narrow for both butt and face places to be driven, it is believed that butt pockets leaving a stump 6 to 7 feet thick will be most satisfactory.

Complete recovery or the removal of all standing pillars that could retard subsidence is essential where the coal is overlain with a strong roof. This is considered of such importance by some companies that every pillar must be removed even if abnormal conditions occasionally make extraction of an individual pillar cost more than may be realized from the coal. This practice is followed to prevent squeezes which might prove more expensive than recovery of single pillars.

In the districts where pillar recovery is practiced, the results sometimes are far from satisfactory. Occasionally, these effects may be attributed to inherently bad roof which will not respond in any marked degree to the mining methods now practiced, but more often the cause may be traced to indifference or inexperience on the part of the management. The more common causes of incomplete success in pillar extraction in the districts where pillar mining is generally practiced are: (1) Irregular alinement of rib lines; (2) conflicting break lines; (3) delay in recovery of pillars; (4) failure to observe the behavior of the roof; (5) improper method of extracting pillars; and (6) incomplete recovery.

Irregular alinement is probably responsible for most of the difficulty experienced in pillar extraction. Every effort should be made by the management to maintain a straight break line. In addition to the methods previously described for maintaining rib alinement, some companies employ pillar inspectors or fall bosses whose principal duty is to see that the break line is kept straight. Whenever possible, the single form of rib line should be used and every effort should be made to avoid the converging V-form.

Conflicting break lines often cause the roof to break over working places, even in solid coal. This generally can be avoided by careful planning and by maintaining a minimum number of rib lines.

Rooms should be driven primarily to develop pillars for recovery rather than as a principal source of output. Overdevelopment in the form of rooms is due generally to the management's being inexperienced in pillar mining and expecting to obtain a large output at low production cost. Rooms where pillars are extracted should be driven just far enough in advance of pillar mining to preclude the possibility of delay along the rib line.

The angle of break in the roof should be determined as early as possible and the break lines should parallel the natural fracture line in the roof. Where this is impracticable, the length of step along the rib line should be regulated to provide the best possible breaks.

The same method of extracting pillars will not be successful in all mines and the management should lose no time in determining the method best adapted to the local conditions. In general, some form of the pocket-and-stump method, open-end method, or combination of the two should be used. Observations in the mines studied seem to indicate that where pillars are mined on the butt cleat the pocket-and-stump method, leaving a stump 7 to 15 feet wide, is most successful; but when places are driven on the face cleats the open-end method has proved best. In a soft coal without pronounced cleat the pocket-and-stump method apparently is better, but if the coal is hard and does not spall along the ribs, the open-end method might prove better.

CONCLUSION

The foregoing discussion deals with engineering practices in mine development. Particular stress is laid on the importance of pillar recovery and on conservation and economy, but the predominating thought has been to present the methods which appear to give promise of greater safety against personal injury from falls of roof or coal.

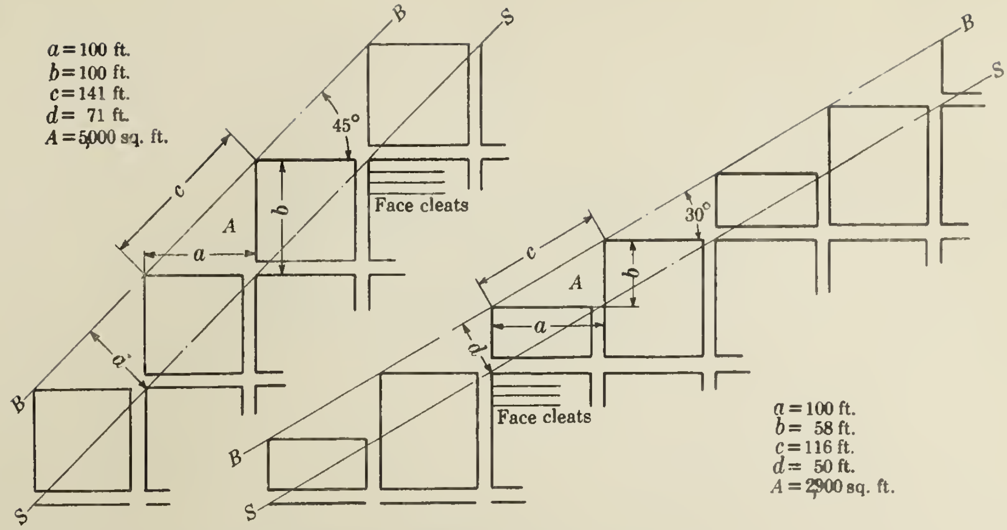

Figure 14.-Comparison of pillar support along 45° and 30° break lines.

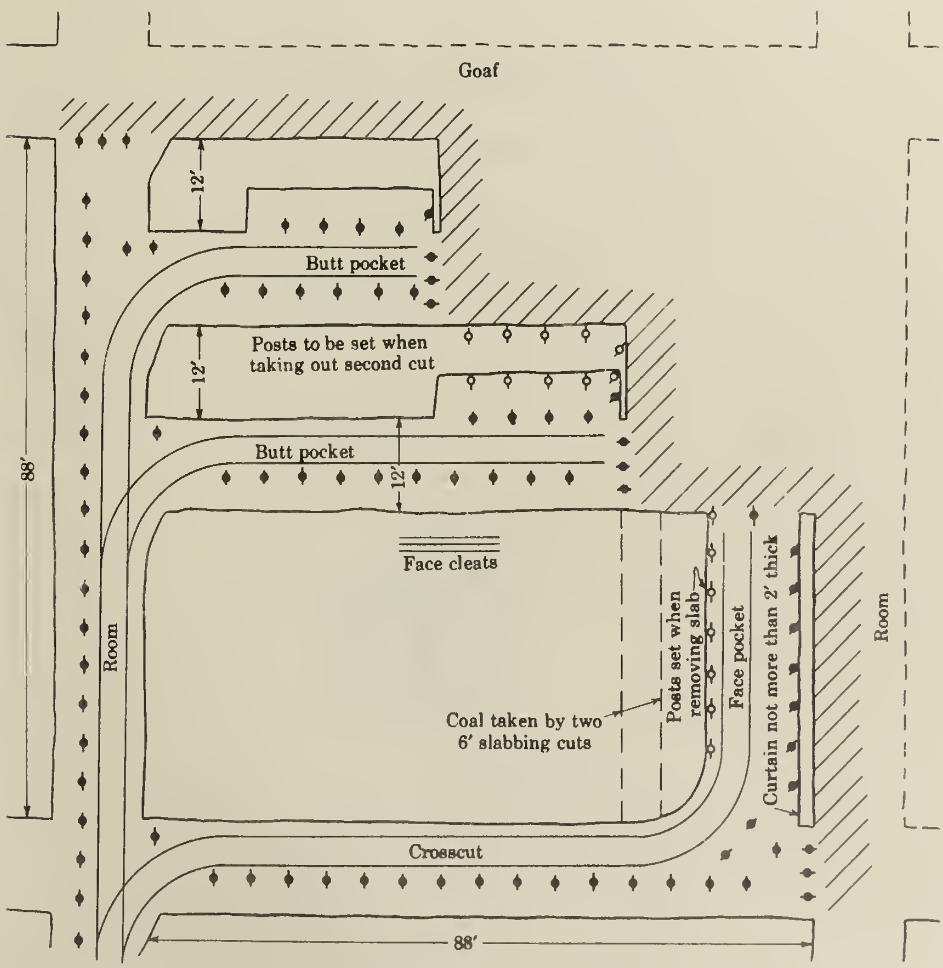

Figure 15.-Mining pillars by a combination of pocket-and-stump and open-end methods.

I. C. 6728 JUNE, 1933

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
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PETROLEUM REFINERIES, INCLUDING CRACKING PLANTS, IN THE UNITED STATES JANUARY 1, 1933

BY

G. R. HOPKINS AND E. W. COCHRANE

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PETROLEUM REFINERIES, INCLUDING CRACKING PLANTS, IN THE UNITED STATES, JANUARY 1, 1933 1/

By G. R. Hopkins2/ and E. W. Cochrane3/

INTRODUCTORY SUMMARY

According to reports received by the Bureau of Mines, Department of Commerce, as of January 1, 1933, there were 505 completed refineries in the United States, an increase of 32 over the total reported at the beginning of 1932, and the largest total reported since January 1, 1926. Of the completed refineries, 372 (74 percent) were operating on January 1, 1933, and 133 (26 percent) were shut down. In addition, 18 refineries were under construction on January 1, 1933,--the largest total in about 10 years. There was a marked similarity between the new construction in 1931 and 1932, as about half of the new plants for both years were in the East Texas field, and many of the others were constructed in other parts of Texas. Comparatively little new equipment was constructed outside of the Mid-Continent in 1932.

The total daily capacity of all the refineries on January 1, 1933, amounted to 3,921,055 barrels, a decrease of 102,273 barrels, or 3 percent, from the previous year. The report for 1932 is the first since 1926 that has not shown an increase in total capacity. In general, the decrease in capacity in 1932 resulted from the cumulative effects of decreased runs of crude to stills, beginning in 1930, plus the continued gain in the percentage yield of gasoline by cracking. Of the total capacity on January 1, 1933, 3,445,118 barrels, or 87.9 percent, represents the capacity of the operating plants; 444,392 barrels, or 11.3 percent, the capacity of the inoperative plants; and 31,545 barrels, or 0.8 percent, the capacity of the plants under construction. Compared with a year ago, these data represent principally an increase in the proportions of the total under construction, balanced by a decrease in the operative ratio.

Texas, with 151 refineries completed or under construction on January 1, 1933, continued to lead all States in number of plants. California, with 61 plants, was second; Oklahoma, with 59, a close third. Although Texas had 27

1/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6728."
2/ Economic analyst, U. S. Bureau of Mines.
3/ Statistical clerk, U. S. Bureau of Mines.

more plants at the end of 1932 than at the beginning, the total daily capacity declined from 939,465 barrels to 914,035 barrels. However, as the total in California showed an even greater decrease in 1932, Texas became, for the first time, the leading State in refining capacity. The list of the States with refineries changed in 1932 as Arizona dropped out and small plants were built in Nebraska and South Dakota.

The major part of the refineries are skimming plants, or plants which process crude oil to three primary products: gasoline, kerosene, and fuel oil. Out of a total of 523 refineries completed and under construction on January 1, 1933, 346, or 66 percent, were skimming plants. This ratio was considerably higher than for the previous year, as nearly all of the new plants built in 1932 were skimming plants. On the basis of capacity, complete plants, or plants which manufacture all of the primary products, the basic constituents of which are present in the crude oils used, outrank skimming plants. On January 1, 1933, 49 percent of the total capacity was represented by complete plants, 35 percent by skimming plants, and 16 percent by the seven other common types.

The survey of January 1, 1933, included for the first time information as to pipe stills used in straight distillation. This canvass showed that there were 685 such stills at refineries in the United States, 569 of which were operating on January 1, 1933. Of the operative pipe stills, 353 were operating on crude, 69 on lube stocks, 75 on pressure distillate, and 72 on "other" oils. The total daily charging capacity of all the pipe stills was 2,959,579 barrels, of which 2,443,556 barrels, or 83 percent, was operative on January 1, 1933. Complete data as to the capacity of the pipe stills operating on crude oil were not obtained, but it is estimated that the total was equivalent to about 50 percent of the crude-oil capacity of the operating refineries.

The 1933 survey also requested information as to reforming units used solely in the cracking of gasoline and naphtha. The canvass showed that the total daily capacity of such units was 45,900 barrels, of which 33,900 barrels was operating on January 1, 1933. Approximately half of the total capacity of the reforming units was represented by cracking units converted to that particular purpose; on the other hand, it did not include the capacity of certain cracking units used in occasional re-forming.

The majority of the new plants added in 1933 were not equipped with cracking units; hence, although the number of refineries having such equipment increased from 207 on January 1, 1932, to 212 a year later, the relative proportion which do not possess such facilities increased.

The total daily capacity of the cracking plants increased steadily through 1931, but in 1932 a small decline was registered. In general, this decrease resulted from the cumulative effects of a material falling off in new construction and a probable increase in the obsolescence rate. The total daily capacity of the completed cracking plants on January 1, 1933, was 1,997,745 barrels; or practically the same as the figure of 1,998,394 barrels recorded in the previous year. The capacity of the plants under construction declined from 48,587 barrels on January 1, 1932, to 33,650 barrels on January 1, 1933, the latter

figure representing the lowest point yet recorded since January 1, 1928. The total capacity of the idle cracking units continued to increase in 1932 and totaled 417,694 barrels on January 1, 1933. The ratio of inoperative equipment to the total capacity is higher with cracking plants than with skimming plants.

As the daily average quantity of fresh feed charged to cracking stills on January 1, 1933, was close to 983,000 barrels, it follows that on that date the cracking units were operating at about 62 percent of their capacity. The operating ratio for the straight-distillation equipment (daily average crude runs divided by operative capacity) on January 1, 1933, was also 62 percent.

The total capacity of the cracking plants in Texas showed a substantial decrease during 1932 but that State continued to far outrank the others in the capacity of such equipment. The rating of the cracking units in New Jersey rose materially in 1932 and that State displaced California as the second ranking State in capacity. A large increase in the number of inoperative cracking units was recorded in California in 1932 but this was more than balanced by a decline in the inoperative capacity in Texas, where many units were dismantled in 1932.

The increasing diversity of terms used to designate the various types of cracking processes and the frequent use of the word "own" in naming processes, indicates a distinct trend toward the construction of cracking plants to suit individual needs. It is estimated that 56 percent of the total daily charging capacity of the cracking units on January 1, 1933 (2,031,395 barrels), was of the unlicensed or "own" group, whereas 44 percent was licensed. A year ago the percentages were 52 for "own" and 48 for the licensed group.

An analysis of the various types indicates that the total daily capacity of the vapor-phase units on January 1, 1933, was at least 75,000 barrels, which, though small in comparison with the liquid-phase capacity, represents an increase in relative importance over that of a year ago.

EXPLANATION OF SYMBOLS AND ABBREVIATIONS USED IN TABLES

Location: The location given is the plant location, which does not always correspond with the office address.

Capacity: Straight distillation.-In general, the capacity of a refinery represents the daily average crude throughput of the plant in complete operation.
Cracking.-The capacity of a cracking plant is usually given as the maximum daily throughput of fresh charging stock.

Status: Op. denotes that the plant was operating on Jan. 1, 1933; similarly, S.d. denotes shut down, and Bldg. denotes building.

Types: Refineries (straight distillation equipment) are differentiated into types according to the products generally made. Eight common types of refineries are given in this survey as follows:

Skimming plant (Skim.)	Skimming and lube (S & L)	Complete plant (Comp.)	Skimming and asphalt (S & A)
Gasoline	Gasoline	Gasoline	Gasoline
Kerosene	Kerosene	Kerosene	Kerosene
Gas oil and fuel oil	Gas oil and fuel oil	Gas oil and fuel oil	Gas oil and fuel oil
	Lubricating oils	Lubricating oils	Asphalt
		Wax, if present in the crude	

Skimming, lube and asphalt, (S, L & A)	Lube plant (Lube)	Asphalt plant (Asph.)	Topping plant (Top.)
Gasoline	Gas oil and fuel oil	Distillates	Tops
Kerosene	Lubricating oils	Gas oil and fuel oil	Distillates
Gas oil and fuel oil		Asphalt	Gas oil and fuel oil
Lubricating oils			
Asphalt			

"Type" used in connection with cracking plants generally represents the manufacturer's designation for the process.

In order to show the connection between this census and the refinery districts as recognized in the refinery statistics of the Bureau of Mines, the following symbols have been attached to the refineries to indicate the divisions to which they have been assigned:

a/ East coast
b/ Appalachian
c/ Indiana, Illinois, Kentucky, etc.
d/ Oklahoma, Kansas, Missouri, etc.
e/ Texas Inland
f/ Texas Gulf coast
g/ Louisiana Gulf coast
h/ Arkansas and Louisiana Inland
i/ Rocky Mountain
j/ California

The footnote 1/ used in the survey indicates that no report was received for January 1, 1933. The information given is taken from the previous survey or from trade sources.

RECAPITULATION OF REFINERIES BY YEARS, 1914-1933, AND BY DISTRICTS AND TYPES, JANUARY 1, 1933 a/

	Number				Capacity (barrels per day)			
	Op.	S.d.	Bldg.	Total	Operating	Shut down	Building	Total
Year:								
Jan. 1, 1914 1/	- - -	- - -	- - -	176	- - -	- - -	- - -	- - -
Jan. 1, 1918	- - -	- - -	- - -	267	- - -	- - -	- - -	1,186,155
Jan. 1, 1919	- - -	- - -	- - -	289	- - -	- - -	- - -	1,295,115
Jan. 1, 1920	2/373	(2/)	99	472	2/1,530,565	(2/)	263,500	1,794,065
Jan. 1, 1921	350	65	44	459	1,794,395	94,405	76,600	1,965,400
Jan. 1, 1922	325	154	30	509	1,854,590	254,610	59,950	2,169,150
Nov. 1, 1924	357	190	8	555	2,480,922	333,410	18,200	2,832,532
Jan. 1, 1925	357	184	6	547	2,489,927	337,910	37,000	2,864,837
May 1, 1925	365	185	4	554	2,511,817	342,025	11,000	2,864,842
Jan. 1, 1926	352	158	2	512	2,562,357	290,610	5,500	2,858,467
Jan. 1, 1927	327	138	7	472	2,834,282	226,725	61,000	3,122,007
Jan. 1, 1928	326	97	5	428	3,036,125	214,255	22,000	3,272,380
Jan. 1, 1929	341	72	14	427	3,325,890	183,650	99,000	3,608,540
Jan. 1, 1930	358	54	8	420	3,634,825	130,760	37,200	3,802,785
Jan. 1, 1931	346	89	10	445	3,706,610	236,075	45,000	3,987,685
Jan. 1, 1932 3/	365	108	6	479	3,624,992	389,616	8,720	4,023,328
Jan. 1, 1933	372	133	18	523	3,445,118	444,392	31,545	3,921,055
Districts, Jan. 1, 1933:								
East coast	23	3	1	27	563,000	39,000	15,000	617,000
Appalachian	45	9	- -	54	149,530	10,750	- - -	160,280
Ind., Ill., Ky., etc.	36	10	- -	46	438,688	12,050	- - -	450,738
Okla., Kans., Mo., etc.	63	16	5	84	449,660	49,980	13,625	513,265
Texas Inland	89	38	6	133	271,570	124,215	2,250	398,035
Texas Gulf coast	14	4	- -	18	497,500	18,500	- - -	516,000
La. Gulf coast	4	3	- -	7	132,000	32,000	- - -	164,000
Ark. and La. Inland	15	4	- -	19	79,550	16,900	- - -	96,450
Rocky Mountain	39	29	6	74	80,485	23,397	670	104,552
California	44	17	- -	61	783,135	117,600	- - -	900,735
Total	372	133	18	523	3,445,118	444,392	31,545	3,921,055
Types, Jan. 1, 1933:								
Skimming (Skim.)	217	111	18	346	955,963	381,342	31,545	1,368,850
Complete (Comp.)	83	4	- -	87	1,894,730	12,500	- - -	1,907,230
Skimming and lube (S&L)	17	4	- -	21	189,000	4,750	- - -	193,750
Skimming and asphalt (S&A)	21	1	- -	22	230,200	22,000	- - -	252,200
Skimming, lube, and asphalt (S, L & A)	2	1	- -	3	22,500	- - -	- - -	22,500
Lube (Lube)	4	1	- -	5	1,575	1,800	- - -	3,375
Asphalt (Asph.)	9	7	- -	16	47,700	7,800	- - -	55,500
Topping (Top.)	18	4	- -	22	103,150	14,200	- - -	117,350
Petrolatum (Petr.)	1	- -	- -	1	300	- - -	- - -	300
Total	372	133	18	523	3,445,118	444,392	31,545	3,921,055

a/ See page 4 for explanation of abbreviations.
1/ From the Bureau of the Census. 2/ Inoperative plants included under operating.
3/ Revised.

RECAPITULATION OF REFINERIES BY STATES (January 1, 1933) a/

State	Number				Capacity (barrels per day)			
	Op.	S.d.	Bldg.	Total	Operating	Shut Down	Building	Total
Alabama	–	1	–	1	– –	6,000	– –	6,000
Arkansas	8	1	–	9	41,500	3,300	– –	44,800
California ...	44	17	–	61	783,135	117,600	– –	900,735
Colorado	5	4	1	10	5,730	1,310	180	7,220
Georgia	1	1	–	2	5,000	4,000	– –	9,000
Illinois	9	2	–	11	128,000	6,700	– –	134,700
Indiana	5	1	–	6	197,000	50	– –	197,050
Iowa	–	1	–	1	– –	1,500	– –	1,500
Kansas	16	–	4	20	150,380	– –	13,125	163,505
Kentucky	8	3	–	11	28,600	800	– –	29,400
Louisiana	11	5	–	16	170,050	39,600	– –	209,650
Maryland	3	–	–	3	55,000	– –	– –	55,000
Massachusetts.	2	–	–	2	48,000	– –	– –	48,000
Michigan	5	4	–	9	18,000	4,500	– –	22,500
Missouri	2	1	–	3	22,000	1,500	– –	23,500
Montana	12	12	–	24	17,350	13,950	– –	31,300
Nebraska	1	–	–	1	60	– –	– –	60
New Jersey ...	6	2	–	8	255,000	35,000	– –	290,000
New Mexico ...	8	2	–	10	5,950	300	– –	6,250
New York	5	–	1	6	40,600	– –	15,000	55,600
Ohio	12	2	–	14	97,610	2,500	– –	100,110
Oklahoma	44	14	1	59	277,220	46,980	500	324,700
Pennsylvania .	37	7	–	44	245,850	8,250	– –	254,100
Rhode Island .	2	–	–	2	6,500	– –	– –	6,500
South Carolina	1	–	–	1	6,500	– –	– –	6,500
South Dakota .	–	–	1	1	– –	– –	40	40
Tennessee	1	–	–	1	58	– –	– –	58
Texas	103	42	6	151	769,070	142,715	2,250	914,035
Utah	2	3	–	5	7,000	1,350	– –	8,350
Virginia	1	–	–	1	1,500	– –	– –	1,500
West Virginia.	6	–	–	6	18,000	– –	– –	18,000
Wyoming	12	8	4	24	44,455	6,487	450	51,392
Total ...	372	133	18	523	3,445,118	444,392	31,545	3,921,055

a/ See page 4 for explanation of abbreviations.

RECAPITULATION OF PIPE STILLS BY DISTRICTS (January 1, 1933) a/

District	Number 1/				Number 2/				Capacity (barrels per day)			
	Op.	S.d.	Bldg.	Total	Crude	Lubes	Press. Dist.	Other oils	Op.	S.d.	Bldg.	Total
East coast	81	9	2	92	45	12	14	10	462,790	40,000	17,000	519,790
Appalachian ...	46	4	–	50	22	15	6	3	139,500	5,500	– –	145,000
Ind.Ill.Ky.,etc.	54	9	1	64	33	5	7	9	322,700	51,500	1,500	375,700
Okla.Kan.Mo.,etc.	85	19	3	107	42	11	17	15	353,450	72,375	14,000	439,825
Texas Inland ..	73	12	1	86	44	7	6	16	165,600	47,900	350	213,850
Tex.Gulf coast.	51	3	–	54	41	5	2	3	310,500	25,000	– –	335,500
La. Gulf coast.	18	6	–	24	16	1	1	0	115,000	27,000	– –	142,000
Ark.& La.Inland	11	1	1	13	10	1	–	–	44,000	13,300	– –	57,300
Rocky Mountain.	35	8	––	43	20	2	6	7	59,356	13,948	– –	73,304
California	115	37	–	152	80	10	16	9	470,660	186,650	– –	657,310
Total	569	108	8	685	353	69	75	72	2,443,556	483,173	32,850	2,959,579

a/ See page 4 for explanation of abbreviations.
1/ Number according to status of operation.
2/ Number in operation according to charging stock.

RECAPITULATION OF CRACKING PLANTS, BY YEARS, 1925-1933, AND BY DISTRICTS AND STATES, JANUARY 1, 1933 a/

	Charging capacity (barrels per day)			
	Operating	Shut down	Building	Total
Year:				
June 1, 1925	690,492	26,200	116,000	832,692
June 1, 1926	844,800	47,690	47,600	940,090
Jan. 1, 1928	1,013,000	253,000	22,000	1,288,000
Jan. 1, 1929	1,194,501	147,923	134,450	1,476,874
Jan. 1, 1930	1,419,200	139,840	149,900	1,708,940
Jan. 1, 1931	1,594,990	244,661	111,130	1,950,781
Jan. 1, 1932	1,603,809	394,585	48,587	2,046,981
Jan. 1, 1933	1,580,051	417,694	33,650	2,031,395
Districts, Jan. 1, 1933:				
East coast	405,879	93,728	- -	499,607
Appalachian	66,114	10,550	1,250	77,914
Ind., Ill., Ky., etc. ..	269,600	53,300	800	323,700
Okla., Kans., Mo., etc..	198,900	64,050	17,900	280,850
Texas Inland	73,150	25,400	7,700	106,250
Texas Gulf coast	301,800	57,100	- -	358,900
Louisiana Gulf coast	47,400	26,200	- -	73,600
Ark. and La. Inland	38,000	12,650	- -	50,650
Rocky Mountain	37,808	12,266	- -	50,074
California	141,400	62,450	6,000	209,850
Total	1,580,051	417,694	33,650	2,031,395
States, Jan. 1, 1933:				
Alabama	- -	3,000	- -	3,000
Arkansas	6,500	9,250	- -	15,750
California	141,400	62,450	6,000	209,850
Colorado	3,350	450	- -	3,800
Georgia	3,600	- -	- -	3,600
Illinois	70,600	15,200	800	86,600
Indiana	148,550	19,500	- -	168,050
Iowa	- -	500	- -	500
Kansas	74,800	32,250	17,900	124,950
Kentucky	10,800	600	- -	11,400
Louisiana	78,900	26,600	- -	105,500
Maryland	58,072	1,500	- -	59,572
Massachusetts	29,500	9,800	- -	39,300
Michigan	6,450	- -	- -	6,450
Missouri	16,000	11,000	- -	27,000
Montana	3,000	2,500	- -	5,500
New Jersey	186,907	54,428	- -	241,335
New Mexico	800	- -	- -	800
New York	19,000	600	- -	19,600
Ohio	50,200	19,000	- -	69,200
Oklahoma	108,100	20,300	- -	128,400
Pennsylvania	133,250	35,950	1,250	170,450
Rhode Island	6,000	- -	- -	6,000
Texas	374,950	82,500	7,700	465,150
Utah	7,400	1,000	- -	8,400
West Virginia	18,664	1,000	- -	19,664
Wyoming	23,258	8,316	- -	31,574
Total	1,580,051	417,694	33,650	2,031,395

a/ See page 4 for explanation of abbreviations.

RECAPITULATION OF CRACKING PLANTS BY TYPES OF PROCESS (January 1, 1933)

Type of process	Charging capacity (barrels per day)			
	Operating	Shut down	Building	Total
Baize	500	- -	- -	500
Black	16,000	- -	- -	16,000
Buerger	4,000	4,000	- -	8,000
Burton	2,558	19,516	- -	22,074
Cross	209,300	32,050	- -	241,350
de Florez	28,700	1,000	- -	29,700
Doherty	26,000	1,500	- -	27,500
Donnelly	- -	2,700	- -	2,700
Dubbs	144,350	56,200	3,650	204,200
Emerson	- -	500	- -	500
Fleming	- -	600	- -	600
Foster-Wheeler	1,000	- -	- -	1,000
Gyro	19,400	1,000	- -	20,400
Holmes-Manley	247,100	3,300	- -	250,400
Isom	26,000	84,500	- -	110,500
Jenkins	26,100	27,700	- -	53,800
Kellogg	7,500	- -	6,000	13,500
Knox	- -	500	- -	500
Koontz	20,000	- -	- -	20,000
Leamon	- -	250	- -	250
Lewis	5,400	- -	- -	5,400
Link	21,800	23,200	- -	45,000
Muehl	- -	1,500	- -	1,500
Ormont	250	1,250	- -	1,500
Own	150,300	36,800	1,500	188,600
Pratt	2,500	- -	800	3,300
Pressure Coke	19,000	- -	- -	19,000
Richmond	27,600	30,300	- -	57,900
Rowsey	2,000	500	- -	2,500
Sinclair Type 600	72,000	5,000	14,500	91,500
Slagter	900	- -	- -	900
Snodgrass	- -	2,200	- -	2,200
Solar	4,000	- -	- -	4,000
True Vapor-phase	4,500	- -	- -	4,500
Trumble	- -	750	- -	750
Tube and Tank	396,843	67,828	- -	464,671
Vapor-phase (miscellaneous)	8,500	4,000	- -	12,500
Winkler-Koch	41,100	2,800	7,200	51,100
Other 1/	44,850	6,250	- -	51,100
Total	1,580,051	417,694	33,650	2,031,395

1/ Includes types designated as continuous pressure, continuous coil, high pressure and cracking coil.

DETAIL BY STATES a/

Company	Location	Straight distillation			Cracking		
		Cap.	Status	Type	Cap.	Status	Type
ALABAMA g/							
1/Coastal Pet. Corp.	Mobile	6,000	S.d.	Skim.	3,000	S.d.	Own
		6,000			3,000		
ARKANSAS h/							
Berry Asphalt Co.	Waterloo	2,000	Op.	Asph.	- -	- -	- -
Henry H. Cross Co.	Smackover	5,000	Op.	Skim.	- -	- -	- -
Houston Oil Co. of Texas	Camden	3,300	S.d.	do.	1,500	S.d.	Dubbs
1/Kettle Creek Refg. Co.	El Dorado	5,000	Op.	do.	2,000	S.d.	do.
Lion Oil Refg. Co.	do.	11,000	Op.	S & A	2,500	Op.	Own
Do.	do.	- -	- -	- -	3,500	S.d.	Burton
Macmillan Pet. Corp.	Norphlet	2,000	Op.	Comp.	- -	- -	- -
1/Ouachita Valley Refg. Co.	El Dorado	2,000	Op.	Skim.	750	S.d.	Dubbs
Root Refg. Co.	do.	12,000	Op.	do.	3,000	Op.	Own
Do.	do.	- -	- -	- -	1,500	S.d.	Dubbs
Simms Oil Co.	Smackover	2,500	Op.	Skim.	1,000	Op.	Cross
		44,800			15,750		
CALIFORNIA i/ *							
Wm. P. Andrews Oil Co.	Newhall	2,000	Op.	Top.	- -	- -	- -
Do.	Signal Hill	2,000	Op.	do.	- -	- -	- -
Associated Oil Co.	Associated	50,000	Op.	Comp.	14,000	Op.	Tube & Tank
Do.	Watson	18,000	Op.	do.	- -	- -	- -
1/R.R. Bush Refg. Co.	Long Beach	3,000	Op.	Top.	- -	- -	- -
1/Caminol Co., Ltd.	Hanford	2,500	Op.	do.	- -	- -	- -
1/ Do.	Santa Fe Springs	3,500	Op.	Skim.	- -	- -	- -
Capitol Crude Oil Co. of Los Angeles	Santa Paula	160	Op.	do.	- -	- -	- -
Chanslor-Canfield Mid-way Oil Co.	Olinda	1,500	S.d.	Top.	- -	- -	- -
Edington Oil & Refg. Co., Ltd.	Long Beach	4,000	Op.	Skim.	- -	- -	- -
Estado Pet. Corp., Ltd.	do.	6,000	Op.	Top.	- -	- -	- -
1/Exeter Oil Co.	Eynes	6,500	Op.	do.	- -	- -	- -
General Pet. Corp. of Calif.	Lebec	12,000	Op.	Skim.	- -	- -	- -
Do.	Los Angeles	50,000	Op.	Comp.	- -	- -	- -
Do.	Torrance	35,000	Op.	Skim.	6,000	Bldg.	Kellogg

a See page 4 for explanation of symbols and abbreviations.
* Data compiled by E. T. Knudsen of the San Francisco office of the U. S. Bureau of Mines.

Company	Location	Straight distillation			Cracking		
		Cap.	Status	Type	Cap.	Status	Type
CALIFORNIA (Cont'd) 1/							
Gilmore Oil Co., Ltd.	Los Angeles	2,200	Op.	S & A	- -	- -	- -
Do.	Roadamite	800	S.d.	Skim.	- -	- -	- -
The Hancock Oil Co. of Calif.	Long Beach	15,000	Op.	Top.	1,200	Op.	Own
1/Hercules Gasoline Co.	Los Angeles	5,500	S.d.	Skim.	1,250	S.d.	Jenkins
Holly Oil Co.	Huntington Beach	1,500	S.d.	do.	- -	- -	- -
Kolingo Refg. Co.	Coalinga	225	Op.	Lube	- -	- -	- -
Lake View Oil & Refg. Co.	Maricopa	2,500	Op.	Skim.	- -	- -	- -
Macmillan Pet. Corp.	Signal Hill	10,000	Op.	Top.	- -	- -	- -
Mohawk Pet. Co.	Bakersfield	2,500	Op.	Skim.	- -	- -	- -
Monarch Refineries, Ltd.	Venice	2,000	Op.	Top.	- -	- -	- -
The Norwalk Co.	Midway	2,500	Op.	do.	- -	- -	- -
Olympic Refg. Co.	Long Beach	6,500	Op.	do.	- -	- -	- -
1/Orange County Refg. Co.	Newport	300	S.d.	Asph.	- -	- -	- -
Paraffine Companies, Inc.	Emeryville	1,200	Op.	do.	- -	- -	- -
1/Petrol Corp.	Los Angeles	5,000	Op.	Skim.	- -	- -	- -
Richfield Oil Co. of Calif.	Bakersfield	2,000	S.d.	do.	- -	- -	- -
Do.	Hynes	55,000	S.d.	do.	11,000	Op.	Cross
Do.	Los Angeles	4,000	S.d.	do.	- -	- -	- -
Do.	Watson	50,000	Op.	Comp.	16,000	Op.	Black
Rio Grande Oil Co.	Vinvale	10,000	S.d.	Skim.	2,000	Op.	Jenkins
Seaside Oil Co.	Summerland	500	S.d.	Asph.	- -	- -	- -
Do.	Ventura	5,000	Op.	S & A	- -	- -	- -
Shell Oil Co.	Coalinga	4,200	Op.	Skim.	- -	- -	- -
Do.	Dominguez	- -	- -	- -	31,500	Op.	Dubbs
Do.	do.	- -	- -	- -	4,500	S.d.	do.
Do.	Martinez	36,500	Op.	Comp.	8,300	Op.	do.
Do.	do.	- -	- -	- -	8,000	S.d.	do.
Do.	Wilmington	58,000	Op.	Comp.	18,400	S.d.	do.
Signal Oil & Gas Co. of Calif.	Hynes	6,000	S.d.	Skim.	- -	- -	- -
1/Southwest Refg. & Sales Co.	Long Beach	1,000	S.d.	Top.	- -	- -	- -
Standard Gasoline Co.	Taft	1,500	Op.	do.	- -	- -	- -
Standard Oil Co. of Calif.	Bakersfield	25,000	Op.	S & A	- -	- -	- -
Do.	El Segundo	100,000	Op.	Comp.	15,000	Op.	Modified Richmond
Do.	do.	- -	- -	- -	21,300	S.d.	do.
Do.	Richmond	100,000	Op.	Comp.	9,000	Op.	Richmond
Do.	do.	- -	- -	- -	9,000	S.d.	do.
The St. Helens Pet. Co., Ltd.	West Whittier	1/4,000	Op.	S & A	- -	- -	- -
Sunset Pacific Oil Co.	Vernon	10,000	S.d.	Skim.	- -	- -	- -

Company	Location	Straight distillation			Cracking		
		Cap.	Status	Type	Cap.	Status	Type
CALIFORNIA (Cont'd) j/							
The Texas Co. (Calif.)	Coalinga	500	S.d.	Top.	- -	- -	- -
Do.	Fillmore	4,000	Op.	Comp.	2,700	Op.	Cross
Do.	Los Angeles	30,000	Op.	Top.	5,500	Op.	Holmes-Manley
1/Triangle Oil & Refg. Co.	Venice	250	Op.	Skim.	- -	- -	- -
Union Oil Co. of Calif.	Avila	10,000	S.d.	do.	- -	- -	- -
Do.	Brea	6,000	S.d.	do.	- -	- -	- -
Do.	Maltha	5,000	Op.	S & A	- -	- -	- -
Do.	Oleum	30,000	Op.	Comp.	- - -	- -	- -
Do.	Santa Paula	900	Op.	Skim.	- -	- -	- -
Do.	Wilmington	55,000	Op.	Comp.	21,000	Op.	Cross
1/Vernon Oil Refg. Co.	Los Angeles	3,000	S.d.	Skim.	- -	- -	- -
Western Oil & Refg. Co.	Wilmington	12,000	Op.	do.	4,200	Op.	Jenkins
Wilshire Oil Co., Inc.	Los Angeles	17,500	Op.	do.	- -	- -	- -
		900,735			209,850		
COLORADO i/							
Berthoud Refg. Co.	Berthoud	30	Op.	Skim.	- -	- -	- -
Continental Oil Co.	Denver	1,500	Op.	do.	1,000	Op.	Cross
Do.	Florence	3,000	Op.	do.	850	Op.	Burton
Do.	do.	- -	- -	- -	450	S.d.	do.
Greasewood Refg. Co.	Orchard	50	S.d.	Skim.	- -	- -	- -
Mancos Dome Refg. Co.	Mancos Creek	60	S.d.	do.	- -	- -	- -
McGarr Pet. Corp.	La Plata County	180	Bldg.	do.	- -	- -	- -
1/Midland Oil Refg. Co.	Denver	1,000	S.d.	do.	- -	- -	- -
1/Mountain States Refg. Co.	Orchard	200	S.d.	do.	- -	- -	- -
Raven Oil & Refg. Co.	Rangely	200	Op.	do.	- -	- -	- -
The Texas Co.	Craig	1,000	Op.	do.	1,500	Op.	Holmes-Manley
		7,220			3,800		
GEORGIA a/							
The Atlantic Refg. Co.	Brunswick	5,000	Op.	S & A	3,600	Op.	Lewis
Mexican Pet. Corp. of Ga.	Savannah	4,000	S.d.	Asph.	- -	- -	- -
		9,000			3,600		
ILLINOIS c/							
Henry H. Cross Co.	Dupo	2,000	Op.	Skim.	- -	- -	- -
Do.	Joliet	3,000	Op.	do.	- -	- -	- -
The Globe Oil & Refg. Co.	Lemont	6,500	Op.	do.	3,000	Op.	Winkler-Koch

Company	Location	Straight distillation			Cracking		
		Cap.	Status	Type	Cap.	Status	Type
ILLINOIS (Cont'd) c/							
Indian Refg. Co., Inc.	Lawrence-ville	16,000	Op.	Comp.	5,000	Op.	de Flo-rez
Do.	do.	- -	- -	- -	6,000	S.d.	Cross
Lincoln Oil Refg. Co.	Robinson	10,000	Op.	S & A	9,600	Op.	Holmes-Manley
Lubrite Refg. Corp.	E. St. Louis	3,500	Op.	Skim.	2,500	Op.	Pratt
Do.	do.	- -	- -	- -	800	Bldg.	do.
Do.	Wood River	5,500	S.d.	Skim.	2,500	S.d.	Dubbs
1/Red River Refg. Co., Inc.	Burnham	1,200	S.d.	Lube	- -	- -	- -
Shell Pet. Corp.	Wood River	45,000	Op.	Comp.	10,000	Op.	Dubbs
Do.	do.	- -	- -	- -	4,500	Op.	True vapor-phase
Do.	do.	- -	- -	- -	4,200	S.d.	Cross
Do.	do.	- -	- -	- -	2,500	S.d.	Dubbs
Standard Oil Co. (Ind.)	do.	22,000	Op.	Comp.	20,000	Op.	Cont. pressure
The Texas Co.	Lockport	20,000	Op.	do.	2,000	Op.	de Flo-rez
Do.	do.	- -	- -	- -	10,000	Op.	Holmes-Manley
Do.	do	- -	- -	- -	4,000	Op.	Pressure coke
		134,700			86,600		
INDIANA c/							
The Bartles-Maguire Oil Co.	E. Chicago	6,000	Op.	Skim.	3,600	Op.	Jenkins
Do.	do.	- -	- -	- -	2,500	Op.	Own
Empire Oil & Refg. Co.	do.	25,000	Op.	Skim.	15,000	Op.	Doherty
Shell Pet. Corp.	do.	27,000	Op.	S & A	13,000	Op.	Dubbs
Sinclair Refg. Co.	do.	40,000	Op.	Comp.	13,000	Op.	Isom
Do.	do.	- -	- -	- -	22,500	Op.	Sinclair Type 600
Do.	do.	- -	- -	- -	19,500	S.d.	Isom
Standard Oil Co. (Ind.)	Whiting	99,000	Op.	Comp.	73,000	Op.	Holmes-Manley
Do.	do.	- -	- -	- -	5,000	Op.	Cont. high pressure
Do.	do.	- -	- -	- -	950	Op.	Whiting vapor-phase
1/Troy Refg. Corp.	Troy	50	S.d.	Skim.	- -	- -	- -
		197,050			168,050		

Company	Location	Straight distillation			Cracking		
		Cap.	Status	Type	Cap.	Status	Type
IOWA d/							
Mona Motor Oil Co.	Council Bluffs	1,500	S.d.	Skim.	500	S.d.	Cross
		1,500			500		
KANSAS d/							
Altitude Pet. Corp.	Chanute	5,000	Op.	Skim.	3,000	Op.	Jenkins
Barnsdall Refineries, Inc.	Wichita	5,000	Op.	do.	1,500	Op.	Dubbs
The Derby Oil Co.	do.	8,000	Op.	do.	3,000	Op.	do.
Do.	do.	- -	- -	- -	1/1,000	S.d.	do.
1/Dickey Refg. Co.	McPherson	3,000	Bldg.	Skim.	1,500	Bldg.	Own
The El Dorado Refg. Co.	El Dorado	4,500	Op.	do.	3,000	Op.	Winkler-Koch
The Globe Oil & Refg. Co.	McPherson	8,000	Bldg.	do.	4,000	Bldg.	do.
Golden Rule Refg. Co.	Wichita	1,000	Op.	do.	1,200	Op.	Jenkins
Independent Oil & Gas Co.	Kansas City	15,000	Op.	do.	2,500	Op.	Dubbs
Do.	do.	- -	- -	- -	2,500	Op.	Own
Kanotex Refg. Co.	Arkansas City	12,000	Op.	Skim.	5,000	Op.	High pressure
Do.	do.	- -	- -	- -	4,500	S.d.	do.
Krueger Oil & Refg. Co.	Natoma	125	Rebldg.	Skim.	- -	- -	- -
National Refg. Co.	Coffeyville	5,500	Op.	Comp.	4,000	Op.	Own
Petroleum Products Co.	Chanute	380	Op.	Skim.	- -	- -	- -
Shell Pet. Corp.	Arkansas City	20,000	Op.	do.	3,200	Op.	Dubbs
Do.	do.	- -	- -	- -	1/2,400	Rebldg.	do.
Sinclair Refg. Co.	Argentine	11,000	Op.	Skim.	5,000	Bldg.	Sinclair Type 600
Do.	do.	- -	- -	- -	13,000	S.d.	Isom
Do.	Coffeyville	12,000	Op.	Comp.	13,000	Op.	do.
Do.	do.	- -	- -	- -	5,000	Bldg.	Sinclair type 600
Skelly Oil Co.	El Dorado	23,000	Op.	S & A	15,000	Op.	Winkler-Koch
Do.	do.	- -	- -	- -	9,000	S.d.	Jenkins
Do.	do.	- -	- -	- -	4,000	S.d.	Skelly-Rittman vapor phase

Company	Location	Straight distillation Cap.	Straight distillation Status	Straight distillation Type	Cracking Cap.	Cracking Status	Cracking Type
KANSAS (Cont'd). d/							
Standard Oil Co. (Ind.)	Neodesha	8,000	Op.	Skim.	3,800	Op.	Cont. pressure
Do.	do.	- -	- -	- -	2,600	Op.	Cracking coil
1/The Sunflower Refg. Corp.	Hutchinson	2,000	Rebldg.	Skim.	750	S.d.	Trumble
Vickers Pet. Co. of Del.	Potwin	4,000	Op.	do.	3,500	Op.	Dubbs
White Eagle Refg. Co.	Augusta	16,000	Op.	do.	9,000	Op.	Own
		163,505			124,950		
KENTUCKY c/							
Aetna Oil Service, Inc.	Louisville	4,000	Op.	Skim.	600	S.d.	Fleming
Ashland Refg. Co.	Leach	3,500	Op.	S & A	1,800	Op.	Dubbs
Bowling Green Refg. Co., Inc.	Memphis Jct.	1,500	Op.	Skim.	- -	- -	- -
1/Glasgow Oil & Refg. Co.	Glasgow	200	S.d.	do.	- -	- -	- -
1/Greenville Refg. Co., Inc.	Greenville	100	S.d.	do.	- -	- -	- -
The Latonia Refg. Corp.	Latonia	8,000	Op.	S & A	5,000	Op.	Tube & Tank
Louisville Refg. Co.	Louisville	4,000	Op.	Skim.	2,000	Op.	Dubbs
Simrall Refg. Corp.	Horse Cave	2,500	Op.	do.	---	- -	- -
1/Somerset Refg. Co.	Somerset	500	S.d.	do.	- -	- -	- -
Stoll Oil Refg. Co.	Louisville	3,000	Op.	Comp.	- -	- -	- -
The Texas Co.	Pryse	2,000	Op.	Skim.	2,000	Op.	Holmes-Manley
		29,400			11,400		
LOUISIANA g/ h/							
Bayou State Refg. Corp.	h/Hosston	800	Op.	Lube	- -	- -	- -
1/Cedar Grove Refg. Corp.	h/Cedar Grove	750	Op.	Skim.	- -	- -	- -
Chalmette Pet. Corp.	g/Chalmette	6,000	Op.	do.	4,000	Op.	Winkler-Koch
Crystal Oil Refg. Corp.	h/Cedar Grove	10,000	S.d.	do.	3,400	S.d.	Jenkins
1/Gupeco Refg. Co., Inc.	h/Alexandria	600	S.d.	Lube	- -	- -	- -
Louisiana Oil Refg. Corp.	h/Bossier City	3,000	S.d.	Asph.	- -	- -	- -
Do.	h/ do.	- -	- -	- -	22,000	Op.	Tube & Tank
Do.	h/Gas Center	15,000	Op.	Skim.	- -	- -	- -
Mexican Pet. Corp. of La., Inc.	g/Destrehan	20,000	Op.	Asph.	- -	- -	- -
Shell Pet. Corp.	g/Norco	16,000	Op.	S & A	3,200	Op.	Dubbs
Sinclair Refg. Co.	g/Meraux	22,000	S.d.	do.	- -	- -	- -
Spartan Refg. Co., Inc.	h/Shreveport	10,000	Op.	Skim.	7,500	Op.	Kellogg

Company	Location	Straight distillation			Cracking		
		Cap.	Status	Type	Cap.	Status	Type
LOUISIANA (Cont'd) g/ h/							
Standard Oil Co. of La.	g/Baton Rouge	90,000	Op.	Comp.	7,400	Op.	Cross
Do.	g/ do.	- -	- -	- -	21,800	Op.	Link
Do.	g/ do.	- -	- -	- -	11,000	Op.	Tube & Tank
Do.	g/ do.	- -	- -	- -	23,200	S.d.	Link
Stanolind Oil & Gas Co.	h/Superior	4,500	Op.	Skim.	3,000	Op.	Cross
The Texas Co.	h/Shreveport	6,500	Op.	Top.	- -	- -	- -
1/U. S. Refg. Co.	g/Southport	4,000	S.d.	Skim.	- -	- -	- -
1/Williams Refg. Co.	h/Shreveport	500	Op.	do.	- -	- -	- -
		209,650			105,500		
MARYLAND a/							
Continental Oil Co.	Baltimore	10,000	Op.	Skim.	4,000	Op.	Cross
Do.	do.	- -	- -	- -	1,500	Op.	Dubbs
Do.	do.	- -	- -	- -	1,500	S.d.	do.
Mexican Pet. Corp.	do.	9,000	Op.	S & A	- -	- -	- -
Standard Oil Co. of N. J.	do.	37,000	Op.	Comp.	52,572	Op.	Tube & Tank
		57,000			59,572		
MASSACHUSETTS a/							
Braintree Oil Processing Co.	E. Braintree	- -	- -	- -	4,500	Op.	Doherty
Cities Service Refg. Co.	do.	20,000	Op.	Comp.	1,800	S.d.	Holmes-Manley
Colonial Beacon Oil Co., Inc.	Everett	23,000	Op.	S & A	25,000	Op.	Tube & Tank
Do.	do.	- -	- -	- -	1/8,000	S.d.	do.
		43,000			39,300		
MICHIGAN c/							
Henry H. Cross Co.	Muskegon	1,000	S.d.	Skim.	- -	- -	- -
Naph-Sol Refg. Co.	do.	2,000	Op.	do.	- -	- -	- -
Old Dutch Refg. Co.	do.	3,000	Op.	do.	- -	- -	- -
1/Peerless Oil Co.	Big Rapids	1,000	S.d.	do.	- -	- -	- -
1/ Do.	Saginaw	1,000	S.d.	do.	- -	- -	- -
The Pure Oil Co.	Midland	3,500	Op.	do.	- -	- -	- -
Roosevelt Oil Co.	Mt. Pleasant	2,500	Op.	S & L	- -	- -	- -
Standard Oil Co. (Ind.)	Zilwaukee	1,500	S.d.	Skim.	- -	- -	- -
White Star Refg. Co.	Trenton	7,000	Op.	do.	2,700	Op.	Controlled coil
Do.	do.	- -	- -	- -	3,750	Op.	Dubbs
		22,500			6,450		

Company	Location	Straight distillation Cap.	Straight distillation Status	Straight distillation Type	Cracking Cap.	Cracking Status	Cracking Type
MISSOURI d/							
Altitude Pet. Corp.	Kansas City	4,000	Op.	Skim.	1,300	S.d.	Burton
Do.	do.	- -	- -	- -	1,500	S.d.	Muehl
Joplin Refg. Co.	Joplin	1/1,500	S.d.	Skim.	1/1,000	S.d.	Jenkins
Standard Oil Co. (Ind.)	Sugar Creek	18,000	Op.	Comp.	16,000	Op.	Holmes-Manley
Do.	do.	- -	- -	- -	7,000	S.d.	Burton
		23,500			27,000		
MONTANA i/							
Aranow Refinery	Kalispell	1,000	Op.	Skim.	500	Op.	Baize
1/ Do.	Havre	400	S.d.	do.	- -	- -	- -
Arro Oil & Refg. Co.	Lewistown	2,000	S.d.	do.	1,000	S.d.	Dubbs
1/Bannatyne Pipeline & Refg. Co.	Collins	750	S.d.	do.	- -	- -	- -
Big Horn Oil & Refg. Co.	Billings	4,000	Op.	do.	- -	- -	- -
Big West Oil Co.	Kevin	1,200	Op.	do.	- -	- -	- -
Conrad Refg. Co.	Conrad	1,000	S.d.	do.	- -	- -	- -
Continental Oil Co.	Lewistown	1,500	Op.	do.	- -	- -	- -
1/Deloraine Refg. Co.	Oilmont	300	S.d.	do.	- -	- -	- -
Dunlap Refinery	Cat Creek	50	Op.	Top.	- -	- -	- -
1/Hart Refineries	Hedgesville	100	S.d.	Skim.	- -	- -	- -
1/ Do.	Missoula	400	S.d.	do.	300	S.d.	Own
Home Oil & Refg. Co.	Great Falls	1,000	S.d.	do.	- -	- -	- -
Eugene Hunt	Winnett	200	Op.	do.	- -	- -	- -
International Refg.Co.	Sunburst	4,500	Op.	do.	2,500	Op.	de Florez
Laurel Oil & Refg. Co.	Laurel	3,000	S.d.	do.	1,200	S.d.	Donnelly
Northwest Stellarene Co., Inc.	Shelby	1,500	Op.	do.	- -	- -	- -
Regal Products Co.	Soap Creek field	50	Op.	do.	- -	- -	- -
1/The Russell Oil Co.	Billings	1,500	S.d.	do.	- -	- -	- -
Do.	Butte	1,000	Op.	do.	- -	- -	- -
Snow Cap Oil Co.	Sunburst	350	Op.	do.	- -	- -	- -
Sunburst Oil & Refg.Co.	Great Falls	1/3,000	S.d.	do.	- -	- -	- -
1/Yale Oil Corp. of S.D.	Billings	2,000	Op.	do.	- -	- -	- -
Do.	Miles City	500	S.d.	do.	- -	- -	- -
		31,300			5,500		
NEBRASKA d/							
Chadron Refinery	Chadron	60	Op.	Skim.	- -	- -	- -
		60			- -		

Company	Location	Straight distillation			Cracking		
		Cap.	Status	Type	Cap.	Status	Type
NEW JERSEY a/							
The Barber Asphalt Co.	Maurer	4,000	Op.	S & A	- -	- -	- -
The Bertrin Pet. Co.	do.	- -	- -	- -	3,000	Op.	Cross
Crew Levick Co.	Petty Island	8,000	Op.	Skim.	- -	- -	- -
Eastern Oil Processing Co.	do.	- -	- -	- -	3,000	Op.	Doherty
Do.	do.	- -	- -	- -	1,500	S.d.	do.
Gulf Refg. Co.	Bayonne	30,000	S.d.	Skim.	1,000	S.d.	de Florez
Middlesex Refg. Co.	Piscataway	5,000	S.d.	do.	500	S.d.	Emerson
Standard Oil Co. of N.J.	Bayonne, etc.	155,000	Op.	Comp.	150,857	Op.	Tube & Tank
Do.	do.	- -	- -	- -	51,428	S.d.	do.
Tide Water Oil Co.	do.	50,000	Op.	Comp.	25,000	Op.	do.
Vacuum Oil Co., Inc.	Paulsboro	20,000	Op.	do.	2,400	Op.	Cross
Do.	do.	- -	- -	- -	2,400	Op.	Tube & Tank
Vapor Phase Oils, Inc.	N. Bergen	- -	- -	- -	250	Op.	Ormont
Warner-Quinlan Co.	Warners	18,000	Op.	S & A	- -	- -	- -
		290,000			241,335		
NEW MEXICO i/							
The Aerex Co.	Bloomfield	100	Op.	Skim.	- -	- -	- -
Basin Refg. Co.	Aztec	50	Op.	do.	- -	- -	- -
Continental Oil Co.	Albuquerque	1,000	Op.	do.	- -	- -	- -
Do.	Artesia	1,500	Op.	do.	- -	- -	- -
Do.	Farmington	1,000	Op.	do.	- -	- -	- -
Malco Refineries, Inc.	Artesia	1,000	Op.	do.	- -	- -	- -
McNutt Oil & Refg. Co.	Brickland	800	Op.	do.	800	Op.	Own
1/Pecos Diamond Refg. Co.	Artesia	250	S.d.	do.	- -	- -	- -
Valley Refg. Co.	Roswell	500	Op.	do.	- -	- -	- -
1/Walker Oil Corp.	Hobbs	50	S.d.	do.	- -	- -	- -
		6,250			800		
NEW YORK a/ b/							
Gulf Refg. Co.	a/Staten Island	15,000	Bldg.	Skim	- -	- -	- -
1/Sawyer Refg. Co.	Wirt	100	Op.	Top.	- -	- -	- -
Sinclair Refg. Co.	b/Wellsville	10,000	Op.	Comp.	4,000	Op.	Sinclair Type 600
Standard Oil Co. of N.Y., Inc.	a/Brooklyn & L. I. City	19,000	Op.	do.	8,000	Op.	Cross
Do.	a/ do.	- -	- -	- -	2,000	Op.	de Florez
Do.	b/Buffalo	5,000	Op.	Comp.	2,000	Op.	Cross
Do.	b/ do.	- -	- -	- -	2,000	Op.	de Florez
Vacuum Oil Co., Inc.	b/Olean	6,500	Op.	Comp.	1,000	Op.	Cross
Do.	b/ do.	- -	- -	- -	600	S.d.	Tube & Tank
		55,300			19,600		

Company	Location	Straight distillation			Cracking		
		Cap.	Status	Type	Cap.	Status	Type
OHIO b/ c/							
Allegheny-Arrow Oil Co.	b/Canton	1,500	S.d.	Skim.	750	S.d.	Dubbs
The Canfield Oil Co.	b/Cleveland	1,000	Op.	Comp.	- -	- -	- -
Gulf Refg. Co. (Del.)	c/Hooven	11,000	Op.	S & A	4,000	Op.	Buerger
Do.	c/ do.	- -	- -	- -	4,000	S.d.	do.
Do.	c/Toledo	14,000	Op.	Skim.	7,000	Op.	Own
Do.	c/ do.	- -	- -	- -	7,000	S.d.	do.
National Refg. Co.	c/Findlay	2,000	Op.	Comp.	1,200	Op.	do.
Do.	b/Marietta	580	Op.	do.	- -	- -	- -
Peninsula Oil Co.	c/Catawba Island	30	Op.	Skim.	- -	- -	- -
The Pure Oil Co.	b/Heath	9,000	Op.	do.	4,000	Op.	Cross
Do.	b/ do.	- -	- -	- -	3,000	Op.	Gyro
Do.	c/Toledo	6,500	Op.	Skim.	6,500	Op.	do.
The Standard Oil Co. (Ohio)	b/Cleveland	20,000	Op.	Comp.	10,000	Op.	Tube & Tank
Do.	c/Lima	7,500	Op.	S & A	4,000	Op.	Solar
Do.	c/ do.	- -	- -	- -	7,000	S.d.	Cross
Do.	c/Toledo	12,000	Op.	Comp.	4,500	Op.	Tube & Tank
The Stellar Refg. Co.	b/Marne	1,000	S.d.	Skim.	250	S.d.	Leamon
Sun Oil Co.	c/Toledo	14,000	Op.	do.	1/6,000	Op.1/	Own 1/
		100,110			69,200		
OKLAHOMA d/							
Anderson-Prichard Refg. Corp.	Cyril	6,000	Op.	Skim.	3,600	Op.	Winkler-Koch
1/Associated Oil Corp.	Allen	8,000	S.d.	Comp.	2,000	S.d.	Jenkins
Barnsdall Refineries, Inc.	Barnsdall	5,000	Op.	do.	4,000	Op.	Cross
Do.	Okmulgee	11,000	Op.	Skim.	4,500	Op.	do.
Bell Oil & Gas Co.	Grandfield	4,500	Op.	do.	2,000	Op.	Dubbs
1/Black Gold Refg. Co.	Oklahoma City	1,500	Op.	do.	- -	- -	- -
1/Century Pet. Co.	do.	2,500	Op.	do.	- -	- -	- -
1/ Do.	W. Tulsa	2,500	S.d.	do.	- -	- -	- -
1/Champlin Refg. Co.	Enid	16,000	Op.	Comp.	3,000	Op.	Winkler-Koch
1/Clay & Peveto	Seminole	500	Bldg.	Skim.	- -	- -	- -
Leslie S. Cole	Oklahoma City	550	Op.	do.	- -	- -	- -
Columbia Refg. Co.	do.	3,500	Op.	do.	- -	- -	- -
Continental Oil Co.	Ponca City	30,000	Op.	Comp.	4,000	Op.	Cross
Do.	do.	- -	- -	- -	4,500	Op.	Dubbs
Do.	do.	- -	- -	- -	4,500	S.d.	do.
Do.	Sapulpa	4,000	S.d.	Skim.	1,500	S.d.	Cross

Company	Location	Straight distillation Cap.	Straight distillation Status	Straight distillation Type	Cracking Cap.	Cracking Status	Cracking Type
OKLAHOMA (Cont'd) d/							
Cushing Refg. & Gasoline Co.	Blackwell	1,700	Op.	Skim.	- -	- -	- -
1/ Do.	Cushing	4,500	Op.	do.	- -	- -	- -
Deep Rock Oil Corp.	do.	10,000	Op.	Comp.	4,000	Op.	Dubbs
Fason Oil Co.	Enid	5,000	Op.	Skim.	2,000	Op.	Jenkins
Empire Oil & Refg. Co.	Cushing	5,000	S.d.	do.	- -	- -	- -
Do.	Okmulgee	4,000	Op.	Comp.	1,000	Op.	Doherty
Do.	Ponca City	12,000	Op.	do.	2,500	Op.	do.
Do.	do.	- -	- -	- -	3,000	Op.	Dubbs
Garber Refinery, Inc.	Garber	4,500	S.d.	Skim.	2,800	S.d.	Winkler-Koch
Gem Refg. Co.	Oklahoma City	200	Op.	do.	- -	- -	- -
The Gilmer Oil Co.	Ringling	1,500	S.d.	do.	- -	- -	- -
The Globe Oil & Refg. Co.	Blackwell	8,000	Op.	do.	4,000	Op.	Winkler-Koch
Gulf States Corp.	Oklahoma City	1/3,000	1/S.d.	1/Skim.	- -	- -	- -
J. C. Huckins	Earlsboro	480	S.d.	Skim.	- -	- -	- -
Illinois Oil Co.	Cushing	3,000	S.d.	do.	1,500	S.d.	Donnelly
1/Imperial Refg. Co.	Ardmore	4,000	S.d.	do.	2,000	S.d.	Dubbs
Independent Oil & Gas Co.	Okmulgee	6,000	Op.	Comp.	2,500	Op.	Dubbs
Indian Territory Ill. Oil Co.	Oklahoma City	1,000	Op.	Top.	- -	- -	- -
Johnson Oil Refg. Co.	Cleveland	6,000	Op.	Skim.	2,000	Op.	Dubbs
Kiowa Refg. Co.	Gotebo	100	Op.	do.	- -	- -	- -
Knox Refg. Co.	Covington	500	Op.	do.	- -	- -	- -
Major Pet. Products Co.	Oklahoma City	1,500	Op.	do.	- -	- -	- -
Marathon Oil Co.	Boynton	2,500	S.d.	Comp.	900	Op.	Slagter
Do.	Bristow	5,000	S.d.	Skim.	- -	- -	- -
Mid-Continent Pet. Corp.	W. Tulsa	40,000	Op.	Comp.	20,000	Op.	Koontz
1/Oklahoma City Refg. Co.	Oklahoma City	2,000	Op.	Skim.	- -	- -	- -
1/J. H. Peacock	do.	2,000	Op.	do.	- -	- -	- -
Pilgrim, Inc.	Kingston	70	Op.	do.	- -	- -	- -
Producers & Refiners Corp.	Tulsa	6,000	Op.	do.	3,000	Op.	Dubbs
Producers Oil Co.	Bristow	1,500	Op.	do.	- -	- -	- -
The Pure Oil Co.	Ardmore	3,000	Op.	do.	2,000	Op.	Dubbs
Do.	Muskogee	9,000	Op.	Comp.	4,000	Op.	Cross
Do.	do.	- -	- -	- -	1,500	Op.	Gyro
Rock Island Refg. Co.	Beckett	5,000	Op.	Skim.	1/3,000	Op.	Winkler-Koch

Company	Location	Straight distillation			Cracking		
		Cap.	Status	Type	Cap.	Status	Type
OKLAHOMA (Cont'd) d/							
Sinclair Refg. Co.	Sand Springs	8,000	Op.	Comp.	4,000	S.d.	Cross
J. M. Singleton	Oklahoma City	600	Op.	Skim.	- -	- -	- -
Sterling Refg. Co.	do.	2,000	Op.	do.	- -	- -	- -
Sun Oil Co. of Del.	Yale	5,000	Op.	do.	- -	- -	- -
Sunray Oil Co.	Allen	6,000	Op.	- -	- -	- -	- -
The Texas Co.	W. Tulsa	14,000	Op.	Comp.	15,000	Op.	Holmes-Manley
Do.	do.	- -	- -	- -	3,500	Op.	Pressure coke
Texas Pacific Coal & Oil Co.	Wynnewood	3,000	Op.	S & A	- -	- -	- -
Tide Water Oil Co. (Okla.)	Drumright	18,000	Op.	Skim.	7,000	Op.	Tube & Tank
Do.	do.	- -	- -	- -	2,000	S.d.	Burton
1/Western Oil Corp.	Beckett	1,500	S.d.	Skim.	1,500	Op.	Jenkins
H. F. Wilcox Oil & Gas Co.	Bristow	5,000	Op.	do.	1,100	Op.	Dubbs
Wirt Franklin Pet. Corp.	Ardmore	4,000	Op.	do.	- -	- -	- -
Yale Oil Corp.	Yale	2,000	Op.	do.	- -	- -	- -
1/York Refg. Co.	Oklahoma City	2,000	S.d.	do.	- -	- -	- -
		324,700			1[illegible]8,400		
PENNSYLVANIA a/ b/							
The Atlantic Refg. Co.	b/Franklin	9,000	Op.	Comp.	6,000	Op.	Cross
Do.	a/Philadelphia	50,000	Op.	do.	20,000	Op.	do.
Do.	a/ do.	- -	- -	- -	10,000	Op.	de Florez
Do.	a/ do.	- -	- -	- -	1,800	Op.	Lewis
Do.	b/Pittsburgh	8,000	Op.	Comp.	4,000	Op.	Cross
James B. Berry Sons Co., Inc.	b/Oil City	2,500	Op.	do.	- -	- -	- -
1/Bradford Oil Refg. Co.	b/Bradford	1,500	Op.	S & L	- -	- -	- -
Bradford-Penn. Refg. Co.	b/Clarendon	500	Op.	Comp.	- -	- -	- -
The Canfield Oil Co.	b/Coraopolis	1,000	Op.	S & L	- -	- -	- -
Carnegie Refg. Co.	b/Carnegie	1,000	Op.	Comp.	- -	- -	- -
Continental Refg. Co.	b/Oil City	1,000	Op.	do.	- -	- -	- -
Crew Levick Co.	b/Titusville	3,000	Op.	do.	- -	- -	- -
Crystal Oil Works	b/Rouseville	1,000	Op.	do.	- -	- -	- -
1/W. H. Daugherty & Son Refg. Co.	b/Petrolia	2,000	Op.	S & L	- -	- -	- -
Franklin Creek Refg. Co.	b/Franklin	1,000	Op.	do.	- -	- -	- -

Company	Location	Straight distillation			Cracking		
		Cap.	Status	Type	Cap.	Status	Type
PENNSYLVANIA (Cont'd) a/ b/							
The Freedom Oil Works Co.	b/Coraopolis) Freedom)	2,500	Op.	Comp.	- -	- -	- -
Gulf Refg. Co.	a/Girard Point	30,000	Op.	S & L	27,000	Op.	Own
Do.	a/ do.	- -	- -	- -	1/2,000	S.d.	do.
Do.	b/Neville Island	6,500	Op.	Skim.	5,200	Op.	do.
Kendall Refg. Co.	b/Bradford	3,500	Op.	Comp.	2,500	Op.	Dubbs
1/A. D. Miller Sons Co.	b/Pittsburgh	1,000	S.d.	S & L	- -	- -	- -
Oil Creek Refg. Co.	b/E. Titus-ville	1,000	Op.	Comp.	- -	- -	- -
1/Penncrude Refg. Co.	b/Kennerdell	400	S.d.	S & L	- -	- -	- -
Pennsylvania Oil Products Refg. Co.	b/Eldred	6,000	Op.	Comp.	1,000	S.d.	Conerty (Snodgrass)
1/Pennsylvania Refg. Co.	b/Karns City	2,000	Op.	do.	- -	- -	- -
1/ Do.	b/Titusville	700	Op.	S & L	- -	- -	- -
The Pennzoil Co.	b/Rouseville	10,000	Op.	Comp.	3,000	Op.	Dubbs
Do.	b/ do.	- -	- -	- -	2,000	S.d.	do.
The Pure Oil Co.	a/Marcus Hook	20,000	Op.	Comp.	8,000	Op.	Cross
Do.	a/ do.	- -	- -	- -	3,000	Op.	Gyro
Pure Penn Refg. Co.	b/Clarendon	1/ 350	S.d.	S & L	- -	- -	- -
Quaker State Oil Refg. Corp.	b/Emlenton	1,650	Op.	Comp.	750	Op.	Dubbs
Do.	b/Farmers Valley	2,000	Op.	do.	1,250	Bldg.	do.
Sinclair Refg. Co.	a/Marcus Hook	20,000	Op.	S & L	17,000	Op.	Sinclair Type 600
Do.	a/ do.	- -	- -	- -	26,000	S.d.	Isom
Starlight Refg. Co.	b/Karns City	1/ 300	1/Op.	1/Petr.	- -	- -	- -
Sun Oil Co.	a/Marcus Hook	45,000	Op.	Comp.	1/25,000	1/Op.	1/Own
Superior Oil Works	b/Warren	1,800	Op.	do.	- -	- -	- -
Swan-Finch Refg. Co.	b/ do.	1,000	S.d.	do.	- -	- -	- -
The Texas Co.	a/Marcus Hook	1,500	Op.	Asph.	- -	- -	- -
Tiona Refg. Co.	b/Clarendon	1,000	Op.	Comp.	- -	- -	- -
Tio Penn Refg. Co.	b/Tiona	500	S.d.	Skim.	- -	- -	- -
Ultra-Penn Refg., Co.	b/Bruin	1,000	Op.	Comp.	1/1,200	S.d.	Snodgrass
United Refg. Co.	b/Struthers	2,000	Op.	do.	- -	- -	- -
Valvoline Oil Co.	b/E. Butler	2,600	Op.	do.	- -	- -	- -
Do.	b/Warren	1,000	S.d.	do.	- -	- -	- -
Vapor Phase Oils, Inc.	b/ do.	- -	- -	- -	1,250	S.d.	Ormont
Waverly Oil Works Co.	b/Coraopolis	4,000	S.d.	Skim.	1,500	S.d.	Cross
Do.	b/Pittsburgh	1,000	Op.	Comp.	1,000	S.d.	do.
Wolverine-Empire Refg. Co.	b/Reno	1,500	Op.	do.	- -	- -	- -
Do.	b/Tidioute	300	Op.	S & L	- -	- -	- -
		254,100			170,450		

Company	Location	Straight distillation			Cracking		
		Cap.	Status	Type	Cap.	Status	Type
RHODE ISLAND a/							
Standard Oil Co. of N.Y., Inc.	E. Providence	1/5,000	Op.	S,L & A	6,000	Op.	Cross
The Texas Co.	Providence	1,500	Op.	Asphalt	- -	- -	- -
		6,500			6,000		
SOUTH CAROLINA a/							
Standard Oil Co. of N.J.	Charleston	6,500	Op.	S,L & A	- -	- -	- -
		6,500			- -	- -	- -
SOUTH DAKOTA i/							
Rex Refg. Co.	Rapid City	40	Bldg.	Skim.	- -	- -	- -
		40			- -		
TENNESSEE c/							
Russell Producing Co.	Bone Camp	58	Op.	Skim.	- -	- -	- -
		58			- -		
TEXAS e/ f/							
Abilene Refinery, Inc.	e/Abilene	330	Op.	Skim.	- -	- -	- -
Alamo Refg. Co.	c/Willow Springs	3,500	S.d.	do.	- -	- -	- -
Allstate Refg. Co.	e/Thrall	2,500	S.d.	do.	- -	- -	- -
American Pet. Co.	f/Norsworthy	4,000	Op.	do.	- -	- -	- -
1/American Refg. Properties	e/Wichita Falls	6,000	S.d.	do.	1,800	S.d.	Dubbs
Archer Refg. Corp.	e/Megargel	1,500	Op.	do.	- -	- -	- -
Arp Refg. Co., Inc.	e/Arp	1,500	Op.	do.	- -	- -	- -
Arrow Refg. & Prod. Co.	e/Overton	2,500	Op.	do.	- -	- -	- -
Atlantic-Pacific & Gulf Refg. Co.	e/Wichita Falls	5,000	S.d.	do.	2,000	S.d.	Jenkins
1/Beacon Oil & Refg. Co.	e/Henderson	3,500	Op.	do.	- -	- -	- -
Bluebonnet Refg. Co.	e/Brownwood	75	S.d.	do.	- -	- -	- -
Bluebonnet Oil Refg. Co.	e/Wickett	5,000	S.d.	Top.	- -	- -	- -
1/Bobrose Oil Refg. Co.	e/Brownwood	1,000	S.d.	Skim.	- -	- -	- -
1/ Do.	e/Luling	2,500	Op.	do.	- -	- -	- -
Burford Oil Co.	e/Pecos	6,000	Op.	do.	- -	- -	- -
Canyon Refg. Co.	e/Kilgore	300	Op.	do.	- -	- -	- -
M. D. Carson Refinery	e/Brady	70	Op.	do.	- -	- -	- -

Company	Location	Straight distillation			Cracking		
		Cap.	Status	Type	Cap.	Status	Type
TEXAS (Cont'd) e/ f/							
Central Refg. Co.	e/Henderson	1/7,500	1/S.d.	1/Skim	- -	- -	- -
Col-Tex Refg. Co.	e/Colorado	10,000	Op.	do.	3,600	Op.	Richmond
Concho Refg. Co.	e/San Angelo	250	Op.	do.	- -	- -	- -
Conroe Refg. Co.	f/Conroe	500	S.d.	do.	- -	- -	- -
1/Constantin Refg. Co.	e/Overton	2,500	Op.	do.	- -	- -	- -
Continental Oil Co.	e/Wichita Falls	5,000	Op.	do.	2,300	Op.	Cross
Cook Refg. Co.	e/Gladewater	350	Bldg.	do.	- -	- -	- -
Cosden Oil Co.	e/Big Spring	10,000	Op.	do.	2,000	Op.	Jenkins
Crown Central Pet. Corp.	f/Pasadena	10,500	Op.	S & L	6,000	Op.	Holmes-Manley
Dale Oil & Refg. Co.	e/Electra	5,000	S.d.	Skim.	- -	- -	- -
Danciger Refineries	e/Bodie	10,000	S.d.	do.	- -	- -	- -
Do.	e/Pampa	6,000	Op.	do.	3,200	Bldg.	Winkler-Koch
Deepwater Oil Refineries, Inc.	f/Houston	3,000	S.d.	S & L	- -	- -	- -
Dixon Creek Oil & Refg. Co.	e/Kingsmill	4,000	Op.	Skim.	- -	- -	- -
1/Dutch Rose Refg. Co.	e/Gladewater	200	Bldg.	do.	- -	- -	- -
1/East Texas Refg. Co.	e/Henderson	6,000	S.d.	do.	- -	- -	- -
1/ Do.	e/Longview	6,000	Op.	do.	3,000	Op.	Own
Empire Oil & Refg. Co.	e/Gainesville	5,000	Op.	do.	2,000	Op.	Dubbs
The Exchange Pet.Corp.	e/Albany	400	Op.	do.	- -	- -	- -
Falcon Lubricants	e/Floresville	50	Op.	Lube	- -	- -	- -
1/Falls Refg. Co.	e/Wichita Falls	2,500	Op.	Skim.	- -	- -	- -
Gladetex Refg. Co.	e/Gladewater	450	S.d.	do.	- -	- -	- -
1/Gratex Refg. Co.	e/Graham	500	Op.	do.	- -	- -	- -
Grayburg Oil Co.	e/San Antonio	1,000	Op.	do.	- -	- -	- -
Great West Refg. Co.	e/Big Spring	7,500	S.d.	do.	- 500	S.d.	Rowsey
1/Gregg Refg. Co.	e/Gladewater	300	Bldg.	do.	- -	- -	- -
Gulf Refg. Co.	e/Fort Worth	6,000	Op.	do.	6,000	Op.	Own
Do.	f/Port Arthur	125,000	Op.	Comp.	30,000	Op.	do.
Do.	f/ do.	- -	- -	- -	20,000	S.d.	do.
Do.	e/Sweetwater	5,000	Op.	Skim.	2,000	Op.	do.
Do.	e/ do.	- -	- -	- -	2,000	S.d.	do.
1/Hartfield Refg. Co., Inc.	e/Kilgore	300	Bldg.	Skim.	- -	- -	- -
Haskell Oil Refg.Corp.	e/Breckenridge	200	Op.	do.	- -	- -	- -
Houston Oil Co. of Tex.	f/Viola	1,000	Op.	do.	- -	- -	- -
Howard County Refg. Co.	e/Big Spring	1,500	Op.	do.	- -	- -	- -
Humble Oil & Refg. Co.	f/Baytown	80,000	Op.	S & L	45,000	Op.	Tube & Tank
Do.	f/ do.	- -	- -	- -	5,600	S.d.	Cross
Do.	e/Breckenridge	1,500	S.d.	Top.	- -	- -	- -

Company	Location	Straight distillation			Cracking		
		Cap.	Status	Type	Cap.	Status	Type
TEXAS (Cont'd) e/ f/							
Humble Oil & Refg. Co.	e/Chilton	4,500	S.d.	Top.	- -	- -	- -
Do.	f/Ingleside	12,000	Op.	Skim.	7,800	Op.	Tube & Tank
Do.	e/McCamey	15,000	S.d.	do.	7,800	S.d.	do.
Do.	e/Neches	5,000	Op.	Top.	- -	- -	- -
Do.	e/San Antonio	4,500	Op.	Skim.	- -	- -	- -
1/Iowa Park Prod. & Refg. Co.	e/Iowa Park	2,500	S.d.	do.	1,250	S.d.	Jenkins
Joco Refinery	e/Stinnett	40	S.d.	do.	- -	- -	- -
1/Johnsonville Refg. Co.	e/Omega	200	S.d.	do.	- -	- -	- -
Kent Refg. Co.	e/Angus	4,000	S.d.	do.	- -	- -	- -
Do.	e/Minerva	2,200	Op.	do.	- -	- -	- -
1/Kilgore Refg. Co.	e/Kilgore	1,500	Op.	do.	- -	- -	- -
1/Lake Refg. Co.	e/Gladewater	500	Op.	do.	- -	- -	- -
1/La Salle Pet. Co.	e/Burkburnett	3,000	S.d.	do.	1,800	S.d.	Jenkins
1/Locke Refg. Co.	e/Gladewater	1,500	S.d.	do.	- -	- -	- -
Macmillan Pet. Corp.	e/Borger	7,000	S.d.	do.	- -	- -	- -
Magnolia Pet. Co.	f/Beaumont	70,000	Op.	Comp.	55,000	Op.	Cross
Do.	e/Corsicana	4,000	Op.	Skim.	- -	- -	- -
Do.	e/Ft. Worth	6,000	Op.	do.	3,500	Op.	Cross
Do.	e/ do.	2,600	S.d.	do.	750	S.d.	Con-trolled coil
Do.	e/Luling	5,000	Op.	do.	3,750	Op.	do.
Do.	f/Magpetco	10,000	S.d.	do.	- -	- -	- -
Marathon Oil Co.	e/Del Rio	2,500	Op.	do.	- -	- -	- -
Do.	e/Fort Worth	5,000	Op.	Comp.	- -	- -	- -
1/Master Pet. Co.	e/Waco	600	S.d.	Skim.	- -	- -	- -
1/Mentone Oil & Refg.Co.	e/Mentone	225	Op.	do.	- -	- -	- -
J. D. Meredith	e/Moran	100	S.d.	do.	- -	- -	- -
Mertzon Refg. Co.	e/Mertzon	125	Op.	do.	- -	- -	- -
Mid-Texas Refg. Co.	e/Eliasville	400	Op.	do.	- -	- -	- -
Misko Refineries, Inc.	e/Mirando City	1,500	Op.	S & L	1,200	Op.	Own
1/Moon Refg. Co.	e/Graham	300	Op.	Skim.	- -	- -	- -
Motor Fuel Products Co.	e/Laredo	1,000	S.d.	do.	1,600	S.d.	Jenkins
Moutray Oil Co.	e/Hawley	750	Op.	do.	- -	- -	- -
Muenster Refg. Co.	e/Muenster	270	Op.	do.	- -	- -	- -
1/Nolting Refg. Co.	e/Sweetwater	700	S.d.	do.	- -	- -	- -
1/Norgold Refg. Co.	e/Olney	800	Op.	do.	- -	- -	- -
O-Bar Refinery	e/Coleman	200	Op.	do.	- -	- -	- -
Octane Oil Refg. Co.	e/Baird	1,500	Op.	do.	- -	- -	- -
Oilton Refinery	e/Oilton	200	Op.	do.	- -	- -	- -
Olney Oil & Refg. Co.	e/Olney	2,000	Op.	do.	1,200	Op.	Own
1/Oriental Oil Co.	e/Dallas	3,500	Op.	do.	- -	- -	- -
1/Overton Refg. Co., Inc.	e/Overton	3,000	Op.	do.	- -	- -	- -
1/Panama Refg. Co.	e/ do.	250	Op.	do.	- -	- -	- -

Company	Location	Straight distillation			Cracking		
		Cap.	Status	Type	Cap.	Status	Type
TEXAS (Cont'd) e/ f/							
Panhandle Refg. Co.	e/Wichita Falls	4,000	Op.	Skim.	1,500	Op.	Dubbs
Paramount Refineries, Inc.	e/San Angelo	2,000	Op.	do.	- -	- -	- -
Pasotex Pet. Co.	e/El Paso	14,000	Op.	do.	4,000	Op.	Dubbs
1/Pecos Refg. Co.	e/Pecos	- -	- -	- -	4,300	Op.	Jenkins
1/Pennant Refg. Co.	e/Kilgore	400	Op.	Skim.	- -	- -	- -
Penn-Tex Refg. & Producing Corp.	e/Echo	450	Op.	do.	- -	- -	- -
Petroleum Conversion Corp.	f/Texas City	- -	- -	- -	500	S.d.	Knox
Phillips Pet. Corp.	e/Borger	30,000	Op.	Skim.	10,000	Op.	Own
1/Phoenix Refg. Co.	e/Eagle Ford	350	Op.	do.	- -	- -	- -
1/ Do.	e/Laredo	500	Op.	Lube	- -	- -	- -
1/ Do.	e/Pettus	600	Op.	Skim.	- -	- -	- -
1/Pilot Point Refg. Co.	e/Pilot Point	500	Op.	do.	- -	- -	- -
1/Pioneer Oil & Refg. Co.	e/Somerset	1,500	Op.	S & L	- -	- -	- -
Primrose Refg. Co.	e/Wichita Falls	2,500	S.d.	Skim.	- -	- -	- -
The Pure Oil Co.	f/Nederland	30,000	Op.	do.	12,000	Op.	Cross
Do.	f/ do.	- -	- -	- -	3,000	Op.	Gyro
1/Republic Oil Refg. Co.	f/Texas City	5,000	Op.	Skim.	3,500	Op.	Winkler-Koch
Richardson Refg. Co.	e/Big Spring	6,000	S.d.	do.	4,400	S.d.	Jenkins
1/Rusk Refg. Co.	e/Overton	3,000	Op.	do.	- -	- -	- -
Sabine Refg. Co.	e/Gladewater	250	Op.	do.	- -	- -	- -
J. Howard Samuell Refinery	e/Coleman	100	Rebldg.	do.	- -	- -	- -
Shamrock Oil & Gas Co.	e/Sunray	2,500	S.d.	do.	- -	- -	- -
Shell Pet. Corp.	f/Houston	29,000	Op.	do.	14,000	Op.	Dubbs
1/Shoreline Refg. Co.	e/Kilgore	300	S.d.	do.	- -	- -	- -
Simms Oil Co.	e/W. Dallas	4,000	Op.	do.	2,000	Op.	Cross
Sinclair Refg. Co.	e/El Paso	5,000	Op.	do.	- -	- -	- -
Do.	e/Fort Worth	4,000	Op.	do.	4,500	Bldg.	Sinclair Type 600
Do.	e/Gladewater	1,500	Op.	Skim.	- -	- -	- -
Do.	f/Houston	31,000	Op.	S & L	28,500	Op.	do.
Do.	f/ do.	- -	- -	- -	5,000	S.d.	do.
Do.	f/ do.	- -	- -	- -	26,000	S.d.	Isom
1/Southern Oil Refg. Co.	e/Reed's Switch	2,500	Op.	Skim.	- -	- -	- -
1/Southland Refg. Co.	e/Olney	650	Op.	do.	- -	- -	- -
Star Refg. & Prod. Co.	e/Fort Worth	1,200	Op.	do.	- -	- -	- -
Stone Oil Co.	f/Texas City	5,000	S.d.	do.	- -	- -	- -
Superior Refg. Co.	e/Tiffin	400	Op.	do.	- -	- -	- -
J. J. & M. Taxman Refg. Co.	e/Wichita Falls	3,500	S.d.	do.	1,500	S.d.	Own

Company	Location	Straight distillation			Cracking		
		Cap.	Status	Type	Cap.	Status	Type
TEXAS (Cont'd) e/ f/							
Taylor Refg. Co.	e/Taylor	4,500	Op.	Skim.	1,000	Op.	Rowsey
Do.	e/Tyler	15,000	Op.	do.	1,000	Op.	do.
The Texas Co.	e/Amarillo	4,000	Op.	do.	1,200	Op.	de Florez
Do.	e/ do.	- -	- -	- -	2,000	Op.	Pressure coke
Do.	e/El Paso	1,500	Op.	Skim.	1,500	Op.	Holmes-Manley
Do.	f/Houston	20,000	Op.	do.	1,000	Op.	Tube & Tank
Do.	f/Port Arthur	60,000	Op.	Comp.	2,000	Op.	de Florez
Do.	f/ do.	- -	- -	- -	90,000	Op.	Holmes-Manley
Do.	f/ do.	- -	- -	- -	4,000	Op.	Pressure coke
Do.	f/Port Neches	20,000	Op.	Asph.	- -	- -	- -
Do.	e/San Antonio	3,000	Op.	Skim.	3,000	Op.	Holmes-Manley
Do.	e/West Dallas	16,000	Op.	do.	2,000	Op.	de Florez
Do.	e/ do.	- -	- -	- -	3,600	Op.	Holmes-Manley
Do.	e/ do.	- -	- -	- -	4,000	Op.	Pressure coke
Texas Oil Products Co.	e/Gladewater	1,000	Bldg.	Skim.	- -	- -	- -
Texas Pacific Coal & Oil Co.	e/Caddo	1,000	Op.	Top.	- -	- -	- -
Do.	e/Fort Worth	3,000	Op.	S & L	1,000	Op.	Cross
1/Texas Pet. Products Co.	e/Somerset	1,500	Op.	Skim.	- -	- -	- -
1/Tonkawa Pet. Corp.	e/Pyote	1,200	S.d.	do.	- -	- -	- -
Trinity Refg. Co.	e/Gladewater	1,000	Op.	do.	- -	- -	- -
1/Unit Refg. Co.	e/Kilgore	250	Op.	do.	- -	- -	- -
1/Valley Consumers Oil Co.	e/Harlingen	1,000	S.d.	do.	- -	- -	- -
1/The Valley Refg. Co.	e/McAllen	500	Op.	do.	- -	- -	- -
1/V. O. E. Refg. Co.	e/Gladewater	500	Op.	do.	- -	- -	- -
Waggoner Refg. Co., Inc.	e/Electra	5,000	Op.	do.	- -	- -	- -
White Star Refg. Co.	e/Cisco	200	Op.	do.	- -	- -	- -
Wickett Refg. Co.	e/Wickett	2,500	Op.	do.	- -	- -	- -
H. F. Wilcox Oil & Gas Co.	e/Pampa	3,000	S.d.	do.	- -	- -	- -
Young Refg. Co.	e/Archer City	250	S.d.	do.	- -	- -	- -
Yuba Oil Co.	e/Nacogdoches	200	S.d.	do.	- -	- -	- -
		914,035			465,150		

Company	Location	Straight distillation			Cracking		
		Cap.	Status	Type	Cap.	Status	Type
UTAH i/							
Diamond Oil Co.	Virgin City	250	S.d.	Skim.	- -	- -	- -
1/Jensen Oil Refg. Co.	Ogden	1,000	S.d.	do.	1,000	S.d.	Own
1/Spartan Oil Co.	Virgin	100	S.d.	do.	- -	- -	- -
Utah Oil Refg. Co.	Salt Lake City	6,000	Op.	Comp.	5,400	Op.	Holmes-Manley
Wasatch Oil Refg.Co.	Woods Cross	1,000	Op.	Skim.	1,000	Op.	Foster-Wheeler
Do.	do.	- -	- -	- -	1,000	Op.	Vapor-phase
		8,350			8,400		
VIRGINIA a/							
The Texas Co.	Norfolk	1,500	Op.	Asph.	- -	- -	- -
		1,500			- -		
WEST VIRGINIA b/							
Carbide & Carbon Chemicals Corp.	S. Charleston	2,000	Op.	Skim.	1,000	Op.	Gyro
Do.	do.	- -	- -	- -	1,000	S.d.	do.
Elk Refg. Co.	Falling Rock	2,000	Op.	S & L	- -	- -	- -
Ohio Valley Refg. Co.	St. Marys	2,000	Op.	Comp.	750	Op.	Dubbs
The Pure Oil Co.	Cabin Creek	3,500	Op.	do.	1,400	Op.	Gyro
Standard Oil Co. of N. J.	Parkersburg	6,000	Op.	do.	13,714	Op.	Tube & Tank
Tri-State Refg. Co.	Kenova	2,500	Op.	Skim.	1,800	Op.	Jenkins
		18,000			19,664		
WYOMING i/							
Bock & Son	Clayspur	90	S.d.	Skim.	- -	- -	- -
California Pet. Corp. (Utah)	Calpet	300	S.d.	do.	- -	- -	- -
Continental Oil Co.	Glenrock	4,000	Op.	do.	1,708	Op.	Burton
Do.	do.	- -	- -	- -	2,016	S.d.	do.
1/Dockery & Pelton	Greybull	200	S.d.	Top.	- -	- -	- -
Eclipse Oil & Refg. Co.	Newcastle	100	Op.	Skim.	- -	- -	- -
Egaso Operating Co.	Osage	750	Op.	do.	750	S.d.	Cross
1/Gillette Refg. Co.	Gillette	100	S.d.	do.	- -	- -	- -
1/Goshen Oil & Refg. Co.	Torrington	250	S.d.	do.	- -	- -	- -
Huber Refinery	Casper	100	Rebldg.	do.	- -	- -	- -
Lusk Oil & Refg. Co.	Lusk	47	S.d.	do.	- -	- -	- -
1/Mountain Refg. Co.	Kemmerer	500	Op.	do.	- -	- -	- -
Northwestern Pet. Co.	Osage	300	Op.	do.	- -	- -	- -
Chas. E. Orchard	Oregon Basin field	50	Bldg.	do	- -	- -	- -

Company	Location	Straight distillation			Cracking		
		Cap.	Status	Type	Cap.	Status	Type
WYOMING (Cont'd) 1/							
Producers & Refiners Corp.	Parco	3,000	Op.	Skim.	4,000	Op.	Dubbs
1/Resolute Oil Corp.	Badger Basin field	50	Op.	do.	- -	- -	- -
Sheridan Refg. Co.	Sheridan	140	Bldg.	do.	- -	- -	- -
Standard Oil Co. (Ind.)	Casper	12,600	Op.	Comp.	6,550	Op.	Vapor-phase
Do.	Greybull	5,000	Op.	S & A	2,500	Op.	Cross
Do.	Laramie	2,500	S.d.	Skim.	3,050	S.d.	Burton
The Texas Co.	Casper	7,000	Op.	do.	5,000	Op.	Holmes-Manley
Do.	do.	- -	- -	- -	1,500	Op.	Pressure coke
Do.	Cody	3,000	S.d.	Skim.	1,500	S.d.	Holmes-Manley
White Eagle Refg. Co.	Casper	6,000	Op.	do.	2,000	Op.	White Eagle controlled coil
Do.	do.	- -	- -	- -	1,000	S.d.	do.
Wyoming Gas & Oil Co.	Osage	155	Op.	Skim.	- -	- -	- -
Wyoming Oils, Inc.	Mills	160	Bldg.	do.	- -	- -	- -
		51,392			31,574		

I. C. 6729 June, 1933

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

MANGANESE

GENERAL INFORMATION

BY

Robert H. Ridgway

I.C.6729,
June, 1933

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MANGANESE[1]

GENERAL INFORMATION

By Robert Ridgway[2]

TABLE OF CONTENTS

INTRODUCTION

This circular outlines salient facts regarding the manganese-ore industry in the United States and the world. It is founded chiefly upon published information available in the literature of the subject. The Bureau of Mines has in its files additional, more detailed data upon many of the topics presented very briefly here, and will endeavor to assist the mineral industries and public by giving further information in response to individual inquiries. The Bureau will welcome comments, criticisms, and contributions of data that may assist in making later additions of this circular more accurate, or in developing more fully its mineral files pertaining to manganese.

In reporting the production of manganese-bearing ores in the United States the following classification has been adopted by the Bureau of Mines: (1) Metallurgical-grade ores, under which have been classified manganese ores containing 35 percent or more of manganese used in the manufacture of iron and steel, ferruginous manganese ore containing 10 to 35 percent of manganese, manganiferous iron ore containing 5 to 10 percent of manganese, and manganiferous zinc residuum; (2) battery ore; (3) fluxing ores, under which are classified ores that may contain a few ounces of silver and are rich enough in manganese to make them valuable chiefly for fluxing purposes in nonferrous smelters; and (4) miscellaneous ores, under which are classified the ores shipped to brick manufacturers, glass makers, and manufacturers of manganese chemicals.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6729."

2 Mining engineer, common metals division, U.S. Bureau of Mines.

In this paper, however, only manganese ore containing more than 35 percent of manganese is considered. For the most part the report is interested in commercial high-grade ores that contain in excess of 45 percent of manganese and are suitable for the manufacture of ferromanganese. The ores mined in the U.S.S.R., India, Brazil, and the Gold Coast are of this quality and make up the bulk of the world consumption and trade in this type of ore. The ore produced in the Sinai peninsula, while of lower grade (30 percent), is of some significance and is given some consideration.

PROPERTIES OF MANGANESE

Manganese is a crystalline, brittle metal, silvery white or grayish white in color with a reddish luster. The specific gravity varies from 7.05 to 8.01, depending on the sample. Its atomic weight is 54.6; melting point, 1,244°D.; boiling point, 1,900°C.; electric conductivity as compared with silver, 1 to 55. Several different allotropic forms of the element are known to exist. To a sodium carbonate bead in an oxidizing flame, manganese imparts a characteristic bluish green color; to a borax bead in an oxidizing flame it gives a purplish or amethystine color. Its chemical properties are similar to those of iron. It oxidizes readily and consequently is never found in nature in the metallic state.

MANGANESE MINERALS

Manganese generally occurs in nature as an oxide, carbonate, or silicate, either alone or in combination with various metals. The oxides pyrolusite and psilomelane are the most important forms. The chief minerals of manganese are described below.

Pyrolusite (MnO_2, generally containing a little H_2O) forms the chief manganese ore. It has a metallic lustre and varies from grayish black to black in color. The mineral is soft and will soil the fingers; its hardness varies from 2 to 2.5 and specific gravity from 4.73 to 4.86. When pure, pyrolusite contains 63.2 percent manganese, but it is rarely found in the pure state; usually it contains small quantities of silica, lime, iron, and barite. The ore usually occurs massive or reniform, occasionally with a fibrous or radiate structure; sometimes it is found powdered or in small grains. The bulk of the Caucasian ore consists of pyrolusite.

Psilomelane (of uncertain composition, perhaps conforming to H_4MnO_5) is often associated with pyrolusite as an ore of manganese. It is iron black to dark steel-gray in color with a dull submetallic luster. It can be distinguished from other manganese oxides by its hardness, which varies from 5 to 6. Specific gravity ranges from 3.7 to 4.7. It is amorphous, the common form being rounded or botryoidal masses, usually with a smooth surface. The content of metallic manganese ranges from 45 to 60 percent. It is often associated with small quantities of barite and potash. Psilomelane is the most abundant mineral in the manganese ores found in India.

Braunite (M_2O_3, $MnSiO_3$, the sesquioxide and silicate of manganese) is invariably associated with silica either mechanically mixed or chemically combined, in amounts up to 8 or 10 percent. It is a brownish or grayish black mineral with a submetallic luster. The ore ranges in hardness from 6 to 6.5, and is frequently too hard to scratch with a knife. The specific gravity varies from 4.75 to 4.82. Braunite, which contains 69 percent metallic manganese, never occurs pure in nature; in ores it is usually associated with other manganese minerals, especially psilomelane. Braunite is a characteristic mineral in the manganese deposits of Arkansas, where it occurs with psilomelane.

Manganite (Mn_2O_3, H_2O, the hydrated sesquioxide) is sometimes known as gray manganese ore. Its color varies from steel-gray to iron-black with a submetallic luster. Its hardness is

4 and its specific gravity 4.3. Manganite frequently alters to pyrolusite, from which it can be distinguished by its hardness and brown streak. Pure manganite contains 62.5 percent manganese. In the United States it is found mainly in California.

Hausmanite ($MnO.Mn_2O_3$) is a brownish black mineral with a dull metallic luster. Its hardness ranges from 5 to 5.5 and its specific gravity from 4.72 to 4.86. The manganese content is 72.1 percent. It is one of the rarer oxides of manganese, but is widely distributed and occurs with other manganese minerals. It is produced artifically by the sintering of carbonates and oxides.

Wad (impure hydrous mixture of oxides of manganese) is an amorphous, earthy, soft, friable mineral with a black or brown color and is commonly known as bog manganese.

Rhodochrosite ($MnCO_3$, carbonate of manganese) is a rose-pink mineral with a pearly luster and has a manganese content of 47.8 percent when pure. It has a hardness of 3.5 to 4.5, being easily scratched with a knife; specific gravity ranges from 3.45 to 3.6. It is a comparatively rare mineral and is usually found associated with other metals in vein deposits. It has been mined as a manganese ore in the French Pyrenees and at Butte, Mont.

Rhodonite ($MnSiO_3$, silicate of manganese) is a rose-colored mineral with a vitreous luster. Its hardness varies from 5.5 to 6.5, being somewhat greater than that of rhodochrosite; specific gravity ranges from 3.4 to 3.68. The content of metallic manganese is 42 percent.

Bementite ($2MnSiO_3.H_2O$, the hydrosilicate of manganese) is a soft mineral, pale grayish yellow to brown in color, with a pearly luster. Its specific gravity is 2.9. It is a common mineral in the deposits in the Olympic Mountains in the State of Washington.

Many other minerals contain varying amounts of manganese, but those just described represent the chief ores of manganese; the two silicates are of very minor importance.

MODE OF OCCURRENCE

Manganese is a common and widely distributed metal. It is very active chemically and is found in many mineral forms. Like iron, it is dissolved out of the crystalline rocks, in which it is almost invariably present, and is redeposited as carbonate, oxide, or hydroxide. The more common mineral compounds are the oxides, carbonates, and silicates. The manganese ores usually consist of a mixture of oxides and are generally of secondary origin, either residual or sedimentary. Of the few bedded deposits known, those at Chiaturi are the best example. Small amounts of carbonates and silicates have been mined for their manganese content in a few places.

The manganese deposits of the United States have been classified according to their form by the U.S. Geological Survey[3] as follows:

1. Stratiform masses:
   (a) Oxides interlayered with sediments or layered volcanic or metamorphic rocks.
   (b) Carbonates or silicates interlayered with sediments or layered volcanic or metamorphic rocks.
   (c) Oxides, as surface bog deposits.
2. Veins and breccia filling:
   (a) Oxides in larger part.
   (b) Carbonates or silicates partly weathered to oxides.
3. Irregular masses:
   (a) Oxides.
   (b) Carbonates or silicates largely weathered to oxides.

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3 Hewett, D. F., Manganese and Manganiferous Ores in 1918: U.S. Geol. Survey, Mineral Resources of the United States, 1918, pt. 1, 1920, pp. 641-642.

4. Aggregates of small oxides in clay or weathered rocks:
   (a) Mainly oxides.
5. Unclassified deposits.

According to their genesis there are two types of deposits: Namely, primary and secondary. The minerals that commonly make up the primary deposits include the following, arranged approximately in order of their importance: Rhodochrosite, siderite, and the related group of carbonates of manganese, iron, calcium, and magnesium; rhodonite; spessartite; tephroite, and other silicates; and alabandite.

Secondary deposits of manganese commonly contain psilomelane, manganite, pyrolusite, braunite, and wad.

WORLD SOURCES

Although the ores of manganese are widely distributed throughout the world, occurrences of large bodies of high-grade ore of commercial importance are limited at present to Brazil, British India, the Gold Coast, the Union of South Africa, and the U.S.S.R. The less important occurrences of high-grade ore include those of Algeria, Austria, Canada, Chile, China, Costa Rica, Cuba, Czechoslovakia, England, Italy, Japan, Mexico, Morocco, Netherland East Indies, New Caledonia, Newfoundland, Panama, Portugal, Puerto Rico, Rumania, Spain, Sweden, Tunisia, and the United States. Lower-grade ore (ferruginous manganese ores) are obtained chiefly from Egypt, the United States, Germany, France, Greece, Italy, and Australia.

Russia

Manganese ores are widely scattered throughout Soviet Russia. However, at the present time the principal producing regions are Chiaturi and Nikopol. A much smaller production comes from the Urals.

Chiaturi.- The deposits at Chiaturi are located on the southern slope of the central part of the Caucasus Mountains about 100 miles east of the Black Sea port of Poti. The mines are centered about the village of Chiaturi in the Province of Kutais, District of Sharopan. The ores at Chiaturi are of sedimentary origin, and the deposits are distinctly stratified. They occur at the base of the Eocene in a regular and continuous horizontal bed ranging from 3 to 10 feet and averaging 7 feet in thickness. The deposit is extensive and exploitation has taken place over 17 square miles, which is divided into seven plateaus by the Kvirila River and its tributaries. The ore deposits are separated by deep valleys and occur at heights of from 500 to 700 feet above the bottoms of the valleys. The ore, which is soft and friable, consists mainly of pyrolusite, but psilomelane and wad are also present. Although production has been continuous since 1879, from the point of view of available tonnage the district still contains large quantities of ore. Estimates as to the extent of reserves differ, but in 1931, Markov estimated that there remained in the beds 72,994,000 metric tons of crude ore which will yield after washing and grading 41,397,000 tons of concentrates with a manganese content of 50 to 53 percent.

Mining is accomplished by underground methods; easy access to the ore is provided by the numerous openings along the outcrops. The ore is transported from the openings to the valley floor either in carts or by aerial tramway. Some of the ore is shipped directly without treatment, while the rest is sorted or washed to increase the manganese content.

The chief source of the Soviet manganese exports is the Chiaturi district which is connected by railroad to the Black Sea port of Poti. The analysis of the ores exported from the Chiaturi district in 1931 has been given[4] as follows:

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4 Metalborse, Von Russischen Manganerz Export: Vol. 22, No. 22, 1932, p. 340.

Analysis of Chiaturi manganese ore exported in 1931, percent

	Manganese	Silica	Phosphorous	Iron
Washed ore....	52	8	0.16	0.95
Ordinary ore	49.6	10.32	.17	1.15

Nikopol.- The most important manganese producing district in southern Russia is at Nikopol, Government of Ekaterinoslav, in the Soviet Republic of Ukraine on the Dnieper River about 100 miles above its entrance into the Gulf of Odessa. There are two main producing regions in the district, one to the northwest of the city of Nikopol in the vicinity of Pokrovsk and the other, known as Vostochny, to the northeast of Nikopol along the Tomakovk river near the villages of Gorodishcke and Krasnogrigorievka. The ore occurs in Oligocene strata as nodules of pyrolusite and psilomelane in a horizontal bed of sandy clay which is near the surface and which ranges from 3 to 5 feet in thickness. The greater part of the ore consists of pyrolusite. The mines are worked by underground methods through shallow shafts. The ore contains from 28 to 33 percent of manganese, but a much higher grade of concentrate is obtained by elimination of about 45 percent of the material mined. Concentration is effected by means of jigs in the eastern district and by dry breaking, screening, and cobbing in the western district. Although the concentrates produced are higher in silica and phosphorous than those produced at Chiaturi, they are good for metallurgical use because of their being clean and coarse.

The following are typical analyses of Nikopol concentrates:

Analysis of Nikopol concentrates, percent

District	Mn	SiO_2	P	Fe
Pokrovsk....................	42-46	12-16	0.2-0.3	0.3-0.7
Vostochny (eastern)	47-52	8-10	0.17	0.3-0.7

The ore reserves at Nikopol have been estimated at different times to be from 7,500,000 tons to 100,000,000 tons. The following table shows the reserves in 1925 as estimated by E. K. Fuchs:

Ore reserves at Nikopol as of 1925

	Area,	Thickness,	Reserves, metric tons			
District	hectares	meters	Actual	Probable	Possible	Total
Pokrovsk..	11,373	0.5-1.5	1,687,000	7,698,000	13,759,000	23,144,000
Vostochny	3,172	0.8-3.5	10,418,000	21,228,000	21,605,000	53,251,000
Total...	14,545	-	12,105,000	28,926,000	35,364,000	76,395,000

Being somewhat closer to the Soviet metallurgical centers, these deposits furnish practically all of the ore consumed in Soviet Russia, and as a consequence are of first importance to the home industry. Some of the ore, however, is exported, most of it going to Germany who draws largely on this deposit for her needs of manganese. The exported ore is of two different types; one type containing 48 percent Mn and the other containing 42 percent Mn. The analysis of the ores exported from the Nikopol district in 1931 has been given[5] as follows:

5 Metalborse, Von Russischen Manganerz Export: Vol. 22, No. 22, 1932, p. 340.

Analysis of Nikopol manganese ore exported in 1931, percent

	Manganese	Silica	Phosphorous	Iron
First grade....	48.43	8.84	0.18	1.30
Second grade.	42.75	14.46	.19	1.90

The district is connected by rail to the Black Sea port of Nikolayev about 200 miles distant.

Other Deposits. There are numerous other deposits in the Urals and Siberia, but only four or five are being exploited and their entire output goes to Russian furnaces. Recent reports indicate that production has been started at the deposits on the Mazul river, a few miles south of Achinsk in Siberia. It is said that the new mines will be able to satisfy the entire demand of the Stalinsk steel plant at Kuznetsk which formerly was to have used Chiaturi ores.

India

There are many known occurrences in India containing large reserves of commercial ore. The principal deposits are in the Balaghat, Bhandara, Chhindwara, and Nagpur districts of the Central Provinces. The producing deposits are of three types described as gondite, kodurite, and lateritoid.[6] In the gondite type the orebodies occur as lenticular masses and bands intercalated with quartzites, schists, and gneisses derived from metamorphosed pre-Cambrian manganese sediments. The orebodies often attain great dimensions, even up to 6 miles in length. The bulk of the ore produced in India, as well as the highest-grade ore, comes from this type of deposit.

The kodurite type of deposit consists of orebodies of irregular size and shape associated with Archean crystalline rocks. Some believe these deposits to be of igneous origin while others suggest that they represent the assimilation of manganese orebodies in an acid magma. They are developed typically in the Vizagapatam district where manganese ore was first mined in India. To date more than 1,000,000 tons have been produced in this district. The ores are usually second and third grade.

The lateritoid deposits consist of irregular masses of iron and manganese ores occurring on the outcrops of pre-Cambrian rocks. The ores are characterized by high iron, low silica, and low phosphorous content and are usually classified as low-grade manganese ores and ferruginous manganese ores.

Fermor gives the locality of the deposits of the various types as follows, the underscored names indicating deposits from which ore has been exported:

(a) Gondite type-

Bihar and Orissa:- Gangpur
Bombay:- Narnkot, Panch Mahals, Chhota Udepur
Central India:- Jhabua
Central Provinces:- Balaghat, Bhandara, Chhindwara, Nagpur, and Seoni

(b) Kodurite type-

Madras:- Ganjam, Vizagapatam

(c) Lateritoid type-

Bihar and Orissa:- Keonjhar. Singhbhum
Bombay:- Dharwar, North Kanara, Ratnagiri
Central Provinces:- Jubbulpore

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6 Fermor, L. L., Manganese: Rec. Geol. Survey of India, vol. 64, 1930, pp. 172-233.

Goa
Madras:- Bellary, Sandur
Mysore:- Chitaldrug, Kadur, Shimoga, Tumkur

Most of the ore is extracted from open pits and is usually of direct shipping grade. Hand sorting is resorted to in order to increase the manganese content, but there is no mechanical concentration of the product. An important factor in the Indian manganese industry is the cost of transportation. Most of the production comes from the Central Provinces and must be hauled long distances by rail to Bombay or Calcutta at considerable cost. New railway lines have been and are being constructed and the harbor of Vizagapatam is being improved in order materially to shorten the rail haul on the bulk of the Indian manganese output.

Although no new areas containing manganese have been discovered in recent years, the reserves of manganese ore are still extensive.

There is some consumption of manganese ore in India, where it is used in the making of ferromanganese and in steel manufacture. In 1929, 47,435 long tons were consumed and in 1930, 46,099 tons were consumed.

Brazil

Manganese ores are widely distributed in Brazil, but the principal deposits of commercial importance are in the State of Minas Geraes. Deposits also occur in the States of Bahia, Matto Grosso, Maranhao, Parana, Pernambuco, Rio de Janeiro, Rio Grande do Norte, Santa Catharina, Cerra, and Sao Paulo. With the exception of Bahia, where some mining has taken place, none of these deposits appears to have been actively mined. Enormous reserves are reported to exist in Matto Grosso, but the distance from seaboard has hampered their development.

The deposits in Minas Geraes are in the central part of the state and include the producing districts of Queluz-Lafayette and Burnier-Ouro Preta. Both districts are connected to Rio de Janeiro by rail, the former being 287 miles north and the latter 317 miles north of this port.

The deposits of Minas Geraes are of two classes, one occurring as beds in metamorphosed sediments of Huronian age associated with iron ore, and the other as lens-shaped bodies in schistose rocks and granite of Laurentian age. Deposits of the second class are residual. The ores of the Burnier district, which are powdered and soft, belong to the first group, while ores of the second group, which are spongy and hard, are found in the Queluz district. The principal minerals are psilomelane, pyrolusite, and manganite. Mining is accomplished chiefly by open-cut methods, although some ore is extracted from underground workings. Practically no machinery is used in mining the ore. Some small washing plants are used but at the largest producing mine no special treatment is given to improve the grade of the ore.

The principal producing mine in Brazil is the Morra da Mina or Merid mine in the Queluz district. The ore, largely psilomelane, is compact and is mined from open cuts. It averages 48 to 51 percent manganese, 5 to 8 percent iron, 3 to 4 percent silica and 0.10 percent phosphorous. Reserves are estimated at 10,000,000 tons of ore. During 1932 this property suspended operations because of lack of market and low prices of ore.

Gold Coast

Although manganese ore is widely distributed in the Gold Coast, only one deposit of major economic importance has been developed to date. It is located along the Sekondi-Kumasi Railway at Dagwin near Taquah in the Gold Coast Colony. The orebodies occur in

ancient sedimentary and metamorphic rocks along a ridge about 2½ miles long. Large amounts of detrital material are also found along the edge of the main ridge. The predominating mineral is psilomelane. All of the production comes from the Nsuta mine of the African Manganese Co., Ltd., at Nsuta, Western Province. The Nsuta mine is probably the largest manganese mine in the world. The ore is removed from open cuts by steam shovels and is hauled to washing plants in steam trains. All of the ore is crushed and washed in log washers, and recovery averages 92 percent. The concentrates of metallurgical grade average 50 to 53 percent manganese, 2 to 4 percent iron, 3 to 7 percent silica, and 0.1 to 0.12 percent phosphorous. The mine also produces chemical-grade ore of good quality. The extent of the reserves has not been ascertained, but there is considered to be 10 million tons of ore in sight.

Regarding the origin of the manganese deposits on the Gold Coast, Junner[7] has this to say, in part:

> The question of the origin of these ores is of considerable economic importance, for if they were formed, as is maintained by certain observers,[8] by the action of descending superficial waters on original sediments containing a relatively small percentage (1 to 20 percent) of manganese, causing the leaching and removal of silica and alumina, etc., from the rock and the leaching and deposition of the manganese in concentrated form, then it is obvious that the extension in depth of workable deposits will be strictly limited.
>
> From his study of the Nsuta ores the writer believes that they represent ancient (pre-Cambrian?) deposits of high-grade manganese ores which in general have not been modified greatly by vadose waters in recent geological times and that where the structural features are favorable the ores are likely to persist in depth.

The extensive reserves and the ease with which the ore of this deposit may be mined and transported to seaboard, makes this source an important factor in the world supply of manganese.

Union of South Africa

The Union of South Africa has never been an important factor in the world's production of manganese, but developments there during the last 5 years warrant its consideration as a potential producer of magnitude. Its advent into the world markets has been delayed by the serious curtailment of steel production but with the return of more normal conditions, production may be of importance.

The principal deposit is in the Postmasburg district in the northern part of the Cape Province about 108 miles west-northwest of Kimberley and about 65 miles northwest of the railhead at Koopmansfontein, from which a branch line has been built to serve the manganese area. The manganese ore occurs as an impersistent layer, or sheet, from 5 to 20 feet in thickness, outcroping at frequent intervals along the crest of a ridge which extends 47 miles in a northerly direction from a point about 5 miles west-northwest of Postmasburg. Psilomelane is the predominant mineral.

The ore, which is hard and contains very little fines, is of high grade, and shipments to industrial consuming centers have been favorably received. It is classified into three grades, the first class containing 51 percent, the second class 48 percent and the third class 44 percent of manganese. The various grades can be recognized and segregated by work-

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7 Junner, Dr. R. N., Report on the Geological Survey Department, 1930-1931, Gold Coast Colony: Aera, 1931, p. 4.

8 Bishop, D. W., and Hughes, W. T., A Contribution to the Geology of the Manganese-Ore Deposits in the Gold Coast Colony and in Ashanti: Trans. Inst. Min. and Met., vol. 39, London, 1930, p. 142-191.

ers at the mine. Although the deposit has been opened up in a number of places, mining is not yet extensive, as there is a considerable amount of rubble ore scattered about the ground. The deposit can be worked cheaply by open-pit methods; the main cost item is likely to be the long rail haul to the seacoast at Durban.

The reserves are extensive and are estimated by some to be as high as 1,000,000,000 tons. The deposits were discovered in 1925, and have been under investigation since 1926. In 1930 the railroad was extended from Koopmansfontein to the deposit. Production began in 1930 but in the last half of 1931 operations were discontinued because of low prices and lack of market.

Egypt

The most important manganiferous area in Egypt is found in the hills of the Um Bogma district, about 19 miles southeast of the Red Sea port of Abou Zenima and about 70 miles south of Suez. The manganese deposits occur irregularly in more or less horizontal bedlike formations below carboniferous limestone. The deposits vary in thickness from 6 inches to 12 feet. They are irregular in size and shape and the ore, which is fairly constant, in combined metal content, varies greatly in relative quantities of iron and manganese, so that the maintenance of a regular grade of ore requires constant attention. Most of the ore is of low grade, averaging 28 to 32 percent of manganese but occasionally running as high as 35 percent. It also contains about 25 percent iron with low percentages of silica, phosphorous, and alumina. This ore is not used extensively in the manufacture of ferromanganese but is consumed largely for the manufacture of basic pig iron in Europe, where it is mixed with the minette iron ores of Alsace-Lorraine.

The ore is mined by underground methods and is shipped without being washed. The ore reserves have been given as in excess of 2,500,000 long tons of assured ore, 3,070,000 tons of probable ore, and 6,274,000 tons of possible ore. Early in 1932 the mines suspended operations until the accumulated production of the last 2 or 3 years has been disposed of.

Cuba

Manganese ore is found in Cuba in Oriente, Santa Clara, Pinar del Rio and Mantanzas provinces, but only in Oriente does it occur in large commercial quantities. Most of the deposits are of low grade and have a high silica content; manganese is usually present as a mixture of pyrolusite, psilomelane, manganite, and wad.

Cuba has never been an important factor in the world market, but its position inside the United States tariff wall has stimulated exploration and research into this island's manganese resources to such an extent that Cuba may become a more potent factor in the United States market. The main development of note is that of the Cuban American Manganese Corporation which controls extensive deposits near Cristo, Province of Oriente, 10 miles north of Santiago on the mainline of the Cuban Railroad. As the ores are not of direct shipping grade, the company has constructed a mill capable of treating 350,000 tons of ore annually and producing 125,000 tons of high-grade concentrates containing 50 percent of manganese. The concentration process involves grinding, flotation, and sintering. The plan also calls for the production of manganese sulphate from the middlings and tailings.

United States

The production of manganese ore in the United States is not important when considering the world output; nor is it significant in magnitude when compared with domestic consumption.

The available reserves of high-grade ore are small and the mining of many deposits on a large scale has been prevented by difficulties of technology or transportation. Manganese-ore deposits of the United States, though small, are widespread. Shipments of manganese ore from 19 States have been reported to the Bureau of Mines during the last decade. The location of these deposits is shown in figure 1, which also contains a diagram depicting the relative importance of the principal producing States during this period. Further details regarding the domestic production are shown in table 1:

Table 1.- Shipments of manganese ore (exclusive of fluxing ore) in the United States by kinds and States for the decade 1922-1931, inclusive, long tons

State	Battery-grade ore	Metallurgical manganese ore	Total manganese ore	Percent of total manganese ore
Alabama	-	4,208	4,208	0.8
Arizona	-	13,939	13,939	2.8
Arkansas	-	33,239	33,239	6.6
California	-	3,493	3,493	.7
Colorado	-	8,443	8,443	1.7
Georgia	-	40,620	40,620	8.1
Idaho	30	6,652	6,682	1.3
Massachusetts	-	35	35	.0
Montana	153,897	170,003	323,900	64.2
Nevada	-	6,631	6,631	1.3
New Mexico	-	15,940	15,940	3.2
North Carolina	-	142	142	.0
Oklahoma	-	33	33	.0
Tennessee	-	4,342	4,342	.9
Texas	-	444	444	.1
Utah	-	165	165	.0
Virginia	8,467	17,231	25,698	5.1
Washington	-	16,275	16,275	3.2
West Virginia	-	29	29	.0
Total	162,394	341,864	504,258	100.0

During the decade ending December 31, 1931, there was produced in the United States 504,258 long tons of manganese ore containing in excess of 35 percent of manganese. Of this total 162,394 tons or 32 percent was produced for consumption by the battery trade. Although there is a small demand for domestic ores in other uses both chemical and metallurgical, virtually all of the remaining 341,864 tons was used by the ferrous metallurgical industries, 256,000 tons or 51 percent of the total having been consumed in the manufacture of ferromanganese alone.

Montana has been the principal source of both domestic battery-grade ore and domestic metallurgical manganese ore, more than 64 percent of the total production coming from that State. The battery-grade ore came entirely from Philipsburg, where it is produced as a magnetic concentrate made from ores mined in this district. Such concentrates average about 72 percent manganese dioxide. The rhodochrosite ore, which contains 35 to 37 percent of

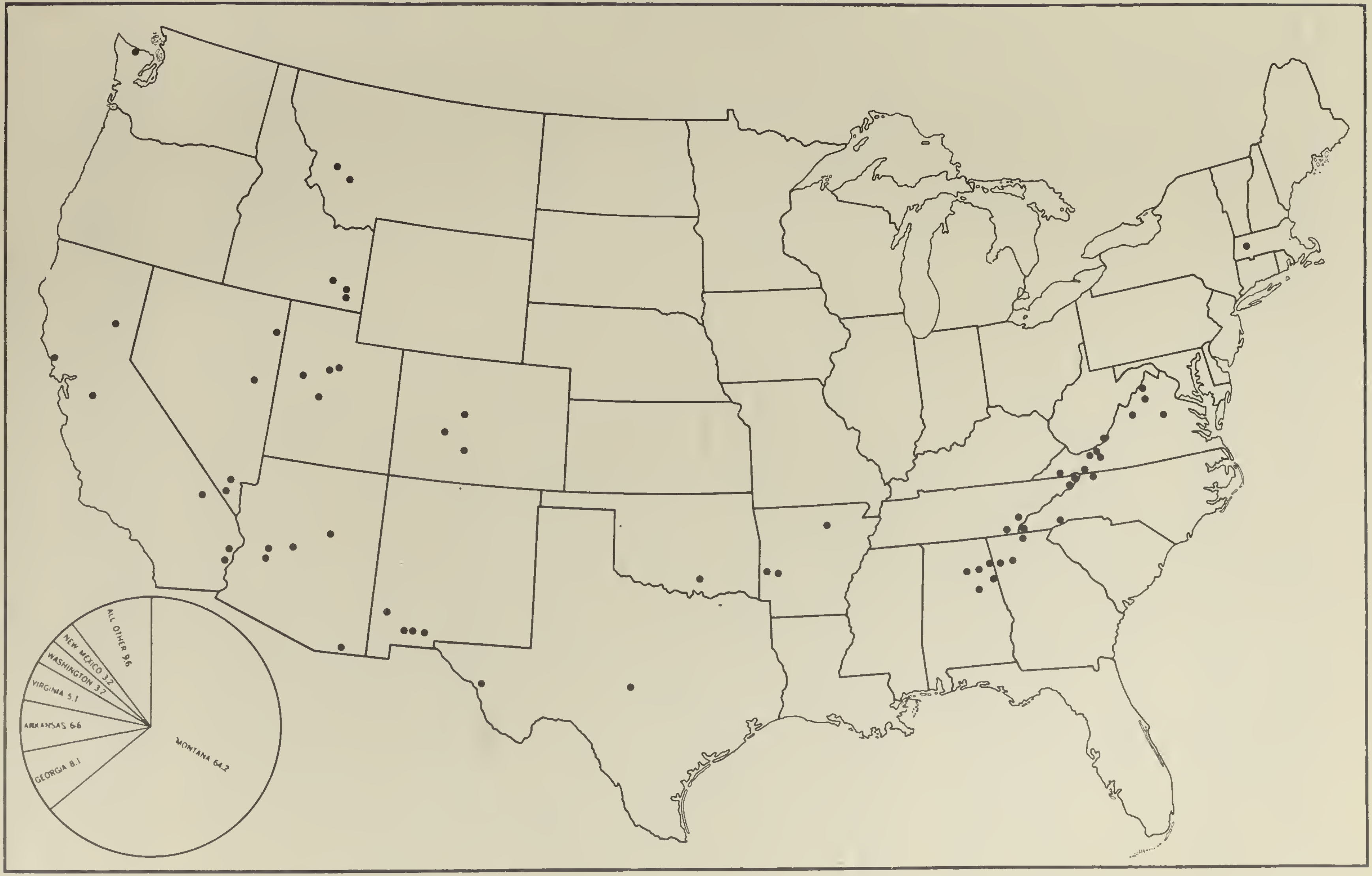

Figure 1.—Localities which have produced manganese ore during the decade 1922–1931 and diagram showing the distribution of production by States during the same decade

manganese, from the Emma mine at Butte constitutes a large part of the metallurgical manganese ore mined in Montana. The rhodochrosite ore, though relatively low in manganese, has been added to the blast furnace charge for the production of ferromanganese. During the last 5 years, however, flotation and nodulizing processes have been developed which render the rhodochrosite ore into a very high-grade product more suitable for the manufacture of ferromanganese. The nodules produced from raw rhodochrosite ore in 1931 contained 58.8 percent of manganese.

Arkansas, Georgia, New Mexico, and Virginia are the other principal producers of manganese ore.

While the reserves of high-grade ore in the United States are small, there is no dearth of lower-grade raw material. The maximum reserve tonnage of high-grade ore was estimated at 3,000,000 long tons in 1925.[9] This figure may be low for a present-day estimate because of several factors, among which the most important are the application of flotation to certain of the rhodochrosite ores in the Butte district which previously had been considered unavailable for ferrous metallurgy and the further development of reserves in the Philipsburg district due to continued exploitation of the mines.

Because of the strategic importance of manganese, more and more attention is being given to the beneficiation of our lcw-grade resources. Perhaps the largest single deposit of manganese in the United States is situated near Chamberlain, S. Dak., where the manganese occurs as manganiferous iron carbonate nodules in a horizontal bed of shale 38 feet thick. The deposit extends over many square miles, and estimates indicate that it contains more than 100,000,000 tons of manganese. The average yield per cubic yard of the shale bed is 164 pounds of nodules, having a manganese content of about 25 pounds. This grade of material is much lower than any thus far worked on a commercial scale either in the United States or elsewhere. The separation of the nodules from the shale matrix presents no serious technical problem, as upon exposure to air the shale dries, cracks, and falls away from the nodules. However, the final separation of the manganese from the nodules into a commercial product involves many technical difficulties. The nodules contain about 16 percent of manganese, 11 percent of iron, and are high in phosphorous (0.381-0.548 percent). The manganese is present as an insomorphous mixture of the carbonates of manganese, iron, and magnesium, and is not amenable to concentration by present known means of ore dressing. Various processes for leaching the nodules for their manganese content are being investigated by Sweet and others but to date such processes have not progressed beyond the laboratory stage.

Another large deposit of low-grade manganese ore exists on the Cuyuna Range in Minnesota. There are two general types of manganese-bearing ores in this district designated by Zappfe[10] as black ores and brown ores. The black ores usually contain 15 to 22 percent of manganese and are low in phosphorous and high in silica. The brown ores usually contain about 10 percent of manganese, are lower in silica, and have a high phosphorous content. The brown ores occur in deposits of large tonnage, whereas the black ores are found in narrow bands containing numerous short lenses usually of small tonnages. The reserves of these two classes of ore have been published in 1927 by Joseph[11] as follows: Brown ores, 20,000,000 tons; black ores, 3,500,000 tons. These figures were based on drill records and other in-

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9 Mining and Metallurgical Society of America, and American Institute of Mining and Metallurgical Engineers. Report of the Sub-Committee on Manganese. International Control of Minerals: New York, 1925, pp. 51-86.

10 Zappfe, Carl, Manganiferous Iron Ores of the Cuyuna District, Minnesota: Trans. Am. Inst. of Min. and Met. Eng., vol. 71, 1925, p. 374.

11 Joseph, T. L., Barrett, E. P., and Wood, C. E., Minnesota Manganiferous Iron Ores in Relation to the Iron and Steel Industry: Trans. Am. Inst. of Min. and Met. Eng., vol. 75, 1927, pp. 304-308.

formation, and are, perhaps, conservative. Regarding reserves on the Cuyuna range, Zappfe[12] has this to say in part: "Estimates of reserves of such ores have never been made with the same accuracy or diligence as has been made for other grades. I showed in 1927 that in the Cuyuna district alone there was a reasonable safe reserve of 44,000,000 tons; and there remained more to be added if the same allowances were to be made here as is customary in the speculations applied to reserves elsewhere. Furthermore, the study revealed that there need be no fear of running short of this kind of ore, insofar as the present known reserves of iron ore were concerned and after allowing even for an increased use of manganiferous ores." In addition to the black and brown classes of ore, there is an enormous amount of manganese-bearing formation in the Cuyuna district which contains more manganese than the ore previously described but which has a very high silica content. This material contains from 40 to 45 percent (dry) of combined manganese (15 percent or more of manganese) and iron and is low in phosphorous, alumina, and moisture. This material has no commercial application at present, but the brown ores and the black ores have been used for some time in blast-furnace burdens in order to furnish manganese for the pig iron used in making basic steel. Some of the ore has been shipped direct and some has been beneficiated by various processes, the most recent of which is the tabling and flotation process worked out at the Merritt mine of the Manganiferous Iron Co. in conjunction with the Bureau of Mines.[13]

None of the ores shipped from the Cuyuna range have been suitable for the manufacture of ferromanganese, but the large reserves have stimulated research in this direction. The Bureau of Mines has developed a pyrometallurgical method for the production of ferromanganese from the brown ores of this district. This process involves three stages:

1. Smelting of manganiferous iron ore, containing 7 to 15 percent of manganese and 31 to 35 percent of iron (with high phosphorus or silica) in the blast furnace to produce a high-manganese (12 percent) pig iron.
2. Open-hearth treatment of this pig iron to produce steel and a high-manganese (55 percent) slag which can be used as an ore.
3. Conversion of the open-hearth slag to ferromanganese in a blast furnace or electric furnace.

This method has been tried out in experimental equipment and ferromanganese has been produced, but it has not been applied to installations of commercial size.

Wilson Bradley has developed a leaching process to recover the manganese from the black ores and the manganese formation high in silica. In this process the ore is crushed and then roasted in a reducing atmosphere at 750°F. in order to reduce the MnO_2 to MnO and the Fe_2O_3 to Fe_3O_4 with a minimum reduction to FeO. The selective roast renders the manganese soluble while leaving the magnetic oxide of iron insoluble so that it can be later recovered from the residue by magnetic concentration as a by-product iron ore. A solution of ammonium sulphate is then used to dissolve the manganese, forming manganese sulphate, from which the manganese is precipitated as manganous hydroxide by the use of ammonia. This process has been worked out successfully on a laboratory scale and was used at a small pilot plant at the Mines Experiment Station at the University of Minnesota. Other experiments in the concentration of manganese have been made but for the most part have not passed beyond the laboratory stage.

While other deposits of manganese exist in various parts of the country, those which have been noted are considered the most important from the national standpoint.

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12 Zappfe, Carl, Manganiferous Iron Ore in Minnesota: Proc. First Ann. Con. Am. Manganese Producers Asso., 1928, pp. 91-97.

13 DeVaney, F. D., and Clemmer, J. B., Concentration Tests on the Manganiferous Iron Ores of the Cuyuna District, Minnesota: Rept. of Investigations 3045, Bureau of Mines, October 1930, pp. 1-9.

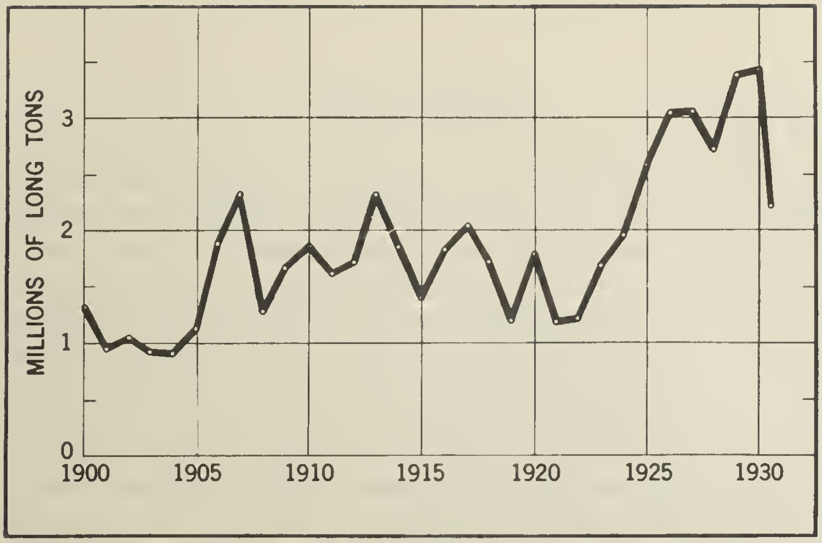

Figure 2.—World production of manganese ore from 1900 to 1931

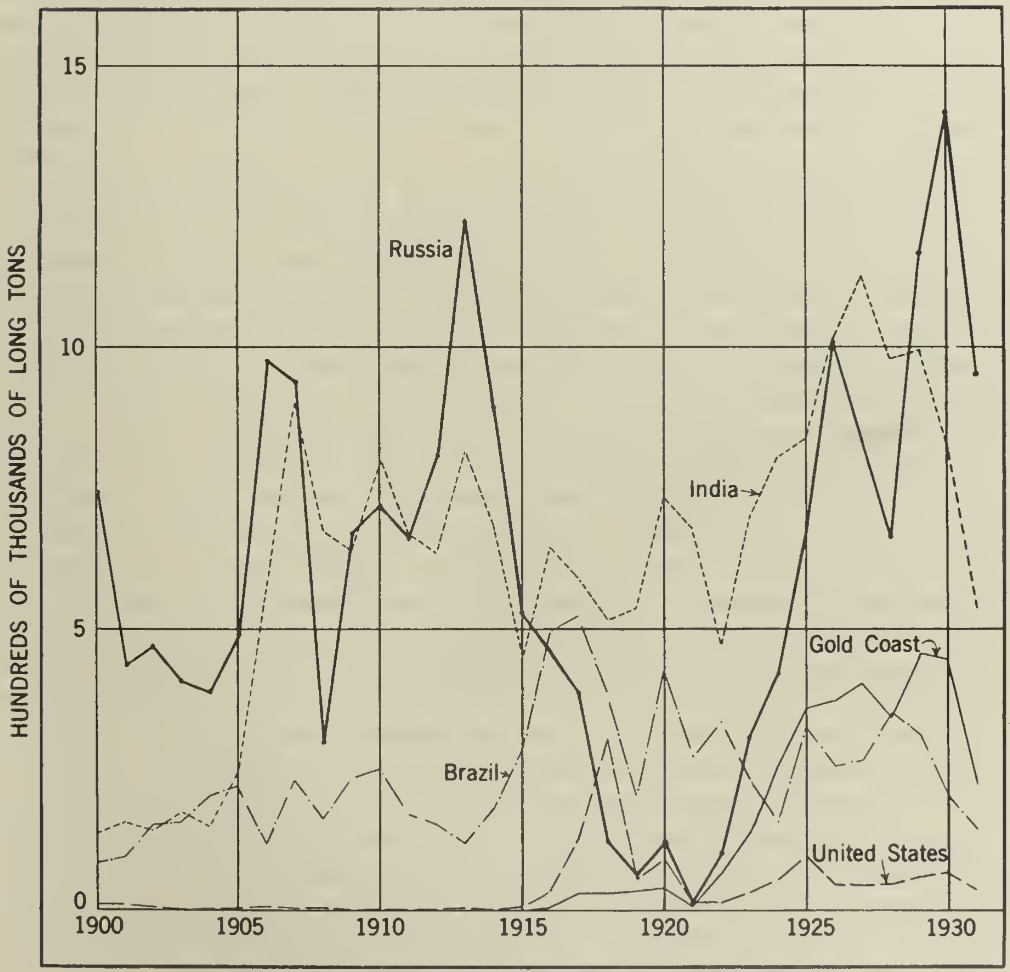

Figure 3.—Production of manganese ore in the main producing countries and the United States from 1900 to 1931

WORLD PRODUCTION

Although manganese has been used since the time of the Egyptians only a relatively small amount was produced up to the second half of the nineteenth century. During the early part of this period it was used as a decolorizing agent in the manufacture of glass. Later in this period it found further outlets in the ceramic industry and in the manufacture of chlorine. The first application of manganese in metallurgy dates from 1839, when Heath applied manganese compounds to the manufacture of steel. The use of manganese in modern metallurgy dates from 1856 when Bessemer invented the unique process of making steel by the pneumatic method. The first ingots produced by this method in England were red-short and cracked or crumbled in the rolls. Mushet's suggestion of adding spiegeleisen to the blown metal to insure malleability while hot was tried successfully. This discovery established the Bessemer process of making steel, and from then on manganese became essential to steel manufacture.

At this time the higher manganese alloys used in steel manufacture were made by the crucible process and were expensive. The increasing production of steel created a demand for cheaper manganese alloys, and as a result the smelting of spiegeleisen and ferromanganese directly from ores in a blast furnace was developed by Pourcel and others in France from 1875 to 1885. The price of high manganese ferro-alloys was thus greatly reduced and consumption increased greatly. This, in turn, led to an increased demand for high-grade ores. Fortunately this demand was satisfied by the discovery and exploitation of large deposits in Russia (1879), British India (1892), and Brazil (1894). In 1899 world production of manganese ore exceeded 1,000,000 long tons for the first time and has since increased to 3,440,000 tons in 1930.

The production of manganese for the world, principal producing countries, and the United States is shown by years from 1900-1931 in table 2. Figures 2 and 3 show graphically the trends of production in the world and in the important producing countries.

According to the shifting sources during the first three decades of the present century, the production of manganese ore can be divided into three very distinct periods. During the first or pre-war period the world market was supplied very largely by Russia, India, and Brazil; during several years over 96 percent of the world's production was mined in these three countries, the greatest competitors being India and Russia. Russia was the largest producer during most of this pre-war period, but India ranked first in 1908, 1910, and 1911. The production of Russia reached a pre-war peak of 1,225,620 tons in 1913, while India's pre-war production reached a peak of 902,291 tons in 1907.

The character of the manganese world industry changed greatly during the World War. Exports from Russia were stopped, causing a considerable decrease in the production of that country. The production in India also decreased owing to difficulties in ocean transport. Curtailment of supplies from these two countries caused an acute shortage in the world markets, and prices increased greatly. Under this stimulus and owing to the comparatively safe ocean transportation to the war-expanded markets in the United States, Brazil increased its output to a peak of 524,439 tons in 1917. In several other countries, including the United States, production was expanded, but largely from lower-grade deposits.

The third or post-war period was characterized by the return to pre-war sources and the addition of the Gold Coast deposits which were discovered in 1916. The ferruginous manganese ores of Egypt have also increased in importance. The restoration of Russia to its pre-war leadership in manganese production was accomplished in 1929 through a large increase in production, while in the past few years India and Brazil have fallen off. The Gold Coast, however, continues to be an important factor due to the high quality of the ore, cheap mining conditions, and a favorable situation regarding transportation.

Table 2.- World production of manganese ores during the present century, showing the outputs of the principal producing countries and the United States, long tons

Year	Russia	India	Brazil[1]	Gold Coast[2]	United States	Other	Total[3]
1900	743,169	139,265	86,735	-	11,771	329,060	1,310,000
1901	435,731	157,736	97,267	-	11,995	249,271	952,000
1902	471,013	144,325	154,775	-	7,477	263,410	1,041,000
1903	407,370	177,821	159,369	-	2,825	180,615	928,000
1904	389,709	150,910	204,971	-	3,146	152,264	901,000
1905	489,845	247,427	220,833	-	4,118	159,777	1,122,000
1906	978,536	571,495	119,415	-	6,921	213,633	1,890,000
1907	941,578	902,291	233,038	-	5,604	240,989	2,323,000
1908	297,526	674,315	163,498	-	6,144	128,517	1,270,000
1909	672,275	642,675	236,971	-	1,544	106,535	1,660,000
1910	721,955	800,907	249,942	-	2,258	92,938	1,868,000
1911	662,981	670,290	171,194	-	2,457	104,078	1,611,000
1912	806,918	637,444	152,424	-	1,664	109,550	1,708,000
1913	1.225,620	815,047	120,368	-	4,048	150,917	2,316,000
1914	892,706	682,898	180,729	-	2,635	94,032	1,853,000
1915	529,000	450,416	284,112	-	9,558	126,914	1,400,000
1916	464,371	645,204	495,184	4,258	31,474	199,509	1,840.000
1917	387,059	590,813	524,439	31,136	129,351	378,202	2,041,000
1918	124,333	517,953	387,175	30,291	305,869	352,379	1.718,000
1919	64,563	537,995	202,476	35,189	54,957	300,820	1,196,000
1920	123,321	736,439	428,855	40,970	94,420	367,995	1,792,000
1921	11,763	679,286	271,340	7,195	13,531	199,885	1,183,000
1922	101,857	474,401	335,325	66,113	13,404	219,900	1,211,000
1923	307,728	695,055	232,107	139,634	31,500	291,976	1,698,000
1924	420,085	803,006	156,714	255,343	56,515	263,337	1,955,000
1925	665,820	839,461	326,585	357,165	98,324	298,645	2,586,000
1926	1,012,365	1,014,928	256,240	371,324	46,258	338,885	3,040,000
1927	830,526	1,129,353	269,174	403,187	44,741	379,019	3,056,000
1928	662,765	978,449	353,970	343,246	46,860	343,710	2,729,000
1929	1,165,187	994,279	311,178	457,932	60,379	409,045	3,398,000
1930	1,421,363	829,946	203,564	445,605	67,035	472,487	3,440,000
1931	[4]950,000	537,844	145,021	223,305	39,242	314,588	2,210,000

[1]Exports from 1900 to 1924. [2]Exports. [3]Partly estimated.
[4]Estimate.

In 1930, world production amounted to 3,440,000 tons, the record production to date. Russia ranked first with a record production amounting to 41 percent of the total. India was second with 24 percent, while the Gold Coast ranked third with 13 percent. Brazil produced 6 percent of the world's total. These four countries accounted for nearly 85 percent of the total output. A more complete list of the sources of manganese ores from 1927 to 1931, inclusive, is shown in table 3:

Table 3.- Manganese ore produced in the principal countries, 1927-1931, long tons

Country[1]	Percentage of manganese	1927	1928	1929	1930	1931
North America:						
Canada (shipments)	-	-	344	269	489	173
Cuba	36-48	799	2,401	957	750	94
Mexico	40+	847	651	245	298	2/
United States-						
Continental (exclusive of fluxing ore)	35+	44,741	46,860	60,379	67,035	39,242
Porto Rico[3]	35+	1,624	1,523	2,316	2,536	2,374
South America:						
Argentina[4]	-	226	139	205	235	-
Brazil	38-50	269,174	353,970	311,178	203,564	145,021
Chile[3]	40-50	7,567	9,047	3,055	6,040	377
Europe:						
Bulgaria	30-45	7	-	-	-	-
Germany	30+	30	207	467	2,312	-
Great Britain	30+	1,031	235	-	-	-
Greece	30+	8,152	1,063	1,575	6,210	350
Hungary	30	16,999	21,817	18,743	8,946	1,114
Italy	30-50	9,610	10,112	9,760	10,465	6,320
Portugal	38-60	483	-	-	-	-
Rumania	42	10,205	30,773	34,485	32,999	18,490
Russia[5]	41-48	830,526	662,765	1,165,187	1,421,363	[7]950,000
Spain	29+	36,288	13,488	17,590	16,553	17,633
Sweden	45+	16,557	15,541	14,378	8,542	8,232
Yugoslavia	42-45	1,940	2,618	3,023	1,515	2,414

[1]In addition to the countries listed Belgium is reported to produce a small quantity of manganese, but statistics of annual output are not available. Czechoslovakia and France report a production of "manganese ore," but as it has been ascertained that the product so reported averages less than 30 percent of manganese and therefore would be considered ferruginous manganese ore under the classification used in this report the output has not been included in the table.
[2]Data not available.
[3]Exports.
[4]Shipments by rail and river.
[5]Year ended Sept. 30.
[7]Estimate.

Table 3.- Manganese ore produced in the principal countries, 1927-1931, long tons-Continued

Country[1]	Percentage of manganese	1927	1928	1929	1930	1931
Asia:						
China[3]	50-55	45,619	42,648	41,219	53,988	21,702
Cyprus	-	16	-	-	-	-
India-						
British	47-52	1,129,353	978,449	994,279	829,946	537,844
Portuguese	42-50+	[3]2,376	-	-	5,390	3,491
Japan	50+	27,124	17,413	18,155	19,279	-
Netherland East Indies	45-56	18,211	24,066	-	16,426	14,311
Portuguese East Indies (Timor)	-	-	-	3,200	-	-
Turkey	-	11,218	60	149	900	1,000
Africa:						
Algeria[3]	-	3,757	1,453	443	1,558	490
Egypt	30+	150,432	135,331	188,454	119,297	100,174
Gold Coast[3]	50+	403,187	343,246	457,932	445,605	223,305
Morocco (French)	40-50+	[3]2,653	[3]2,110	-	6/	11,320
Northern Rhodesia	41-50	694	1,792	1,849	873	1,467
Tunisia	30-43	2,024	2,165	197	-	-
Union of South Africa	40-60	1,279	-	9,202	144,995	100,290
Oceania:						
Australia-						
New South Wales	-	1,202	167	233	125	-
Queensland	50+	242	-	-	-	-
South Australia	-	-	-	-	-	13
Victoria	50+	15	-	-	4/	-
Western Australia[3]	47+	30	-	80	-	-
New Caledonia[3]	-	128	-	-	-	-
New Zealand[3]	52+	5	-	-	2	-
	-	3,056,000	2,729,000	3,398,000	3,440,000	2,210,000

[1]In addition to the countries listed Belgium is reported to produce a small quantity of manganese, but statistics of annual output are not available. Czechoslovakia and France report a production of "manganese ore," but as it has been ascertained that the product so reported averages less than 30 percent of manganese and therefore would be considered ferruginous manganese ore under the classification used in this report the output has not been included in the table.

[3]Exports.

[6]Estimate included in total.

USES OF MANGANESE ORE

The uses of manganese ore may be considered under two broad classifications: (1) metallurgical, and (2) chemical.

Metallurgical Uses

Although manganese ore is consumed in both the ferrous and nonferrous metallurgical industries, the bulk of the world's output finds outlet in the manufacture of iron and steel. Various estimates indicate that more than 90 percent of the world's consumption is in the ferrous metallurgical industry. Most of the manganese ore entering this field is used in the making of ferromanganese and spiegeleisen, the forms in which manganese is usually added to steel. These two alloys of iron, carbon, and manganese have the following composition:

	Ferromanganese, percent	Spiegeleisen, percent
Manganese	78-82	18-22
Iron	8-15	70-80
Silicon	0.5-1	1
Carbon	5-7	5-6
Phosphorus	0.1-0.3	0.15
Sulphur, less than	0.03	0.05

Silicomanganese and silicospiegel are used in certain grades of steel and may replace ferromanganese and spiegeleisen. The usual requirements, as to analysis, for these two alloys are:

	Silicomanganese, percent	Silicospiegel, percent
Manganese	55-70	20-50
Iron	5-20	43-67
Silicon	25	4-10
Carbon	0.35	1.5-3.5

Considerable manganese ore is also added to the pig-iron blast-furnace charge when the iron-ore burdens are deficient in manganese, especially when the pig iron is required for steel-making processes.

Under present technology, the use of manganese is indispensable in the manufacture of steel, where it is used to improve blast-furnace operation and to offset the harmful effects of oxygen and sulphur. It is used not only in the process of making plain-carbon steel, but also as an alloying constituent because of certain properties it imparts to alloy steels. Manganese steels are utilized in the manufacture of plates, shapes, structural bars, open-hearth rails, spring steels, car wheels, tires, axles, and for many other purposes where toughness and resistance to abrasion are required. Some manganese is also used in foundry work.

The greater part of the manganese ore mined is used in the manufacture of ferromanganese, either in the blast furnace or in the electric furnace. In the process virtually all of the iron and phosphorus in the charge enters the resultant alloy. Consequently, the usability of the ore depends on the ratio of manganese to iron and on the phosphorus content. The initial requirement of manganese ore is that the ratio of manganese to iron should be about 10 to 1; so that, allowing for normal loss from 12 to 18 percent in the total manganese, the ratio of manganese to iron in the alloy may be maintained. Silica above 8 percent makes ferrograde ores undesirable. The following analyses[14] (table 4) are typical of the general run of foreign ores used in this country for the manufacture of ferromanganese.

14 Langhorne, M. D., Grades and Character of Ore Desired by the Market: Proc. First Ann. Conv. Am. Manganese Producers Assoc., 1928, pp. 124-127.

Table 4.- Analysis of foreign manganese ores consumed in the United States

Source of ore	Analysis (natural state), percent					
	Manganese	Iron	Silica	Alumina	Phosphorus	Bases
Gold Coast......	46.7	6.25	3.15	4.1	0.14	1.50
India...............	50.51	6.22	8.28	1.83	.09	1.50
Brazil...........	48.70	4.40	4.60	2.95	.12	2.50
Chile.............	48.70	.73	9.40	1.69	.008	-
Russia...........	49.19	.64	7.85	1.57	.22	-

Fine and soft ores are particularly objectionable in blast-furnace practice as they impede the blast and cause loss as flue dust. Hard ore in medium-sized lumps is desired, so that the weight of the charge may be sustained without breakage.

Much lower grade ores are used in the manufacture of spiegeleisen and manganiferous pig iron. Such ores have been classified by the Bureau of Mines as ferruginous manganese ores (Mn content 10 to 35 percent) and manganiferous iron ores (Mn content 5 to 10 percent). Large quantities of ore containing from 2 to 5 percent of Mn are consumed largely in the manufacture of basic open hearth pig. The combined iron and manganese content of such ores, however, should be about 50 percent.

Manganese is also used in the formation of alloys of copper, zinc, aluminum, and other metals. Manganese ores have also been used for fluxing in nonferrous smelting operations.

That there is no extensive use of pure or nearly pure metallic manganese is due in part to the difficulties and high cost of producing the metal. It is consumed largely for experimental purposes.

Chemical Uses

Although the greater part of the production of manganese ore is consumed in the metallurgical industries, many valuable and growing applications of this raw material are found in the chemical industries.

Dry-cell Manufacture.- Probably the most extensive chemical use of manganese ore is in the manufacture of dry cells in which the function of the manganese dioxide is that of a depolarizer. Manganese dioxide used for this purpose should have a high content of available oxygen with minimum iron and be comparatively free from metals such as arsenic, copper, nickel, or cobalt, which are electronegative to zinc. The physical properties of the oxide are important. The material should be porous and moderately hard. Most of the imported material used in dry-cell manufacture in this country contains in excess of 80 percent of MnO_2, but since the war extensive use has been made of domestic material containing about 72 percent of MnO_2 because of its good physical properties. In the United States in the recent past the use of MnO_2 for this purpose has declined with the advent of radio sets which operate on ordinary house current.

Glass Industry.- Another outlet taking advantage of the oxidizing power of manganese dioxides is in the glass and ceramic industry. Most of the sand used in these industries contains iron which imparts a greenish tinge to the glass. The available oxygen of the manganese dioxide converts the greenish salt into the nearly colorless salt, while the slight pink tint imparted by the manganese salt is complimentary to the bluish color of the oxidized iron. Fine glassware is almost entirely decolorized by the addition of manganese dioxide. The quantity of manganese dioxide used varies from 2 to 15 pounds per 1,000 pounds of sand, depending on the character of the glass, the method of its manufacture, the iron content of

the raw material, the character of the manganese ore used, and the temperature at which the glass is made. Wad, the earthy variety of manganese oxide, is used in conjunction with other oxides in this industry. High-grade ores with a small iron content are required for this trade, although siliceous pyrolusite is not objectionable.

Paint and Varnish Driers.- Manganese compounds are used extensively as driers in the preparation of varnish and paint, due to their catalytic properties. The manganese compounds used are natural and artificial manganese dioxide, manganese hydrate, borate, chloride, sulphate, resinate, linoleate, oxalate, and other salts. Manganese ore required for this use must be of relatively high grade.

Pigments and Dyeing Materials.- Manganese compounds are used in coloring glass, pottery, tiles, and brick; in calico printing and dyeing; and for certain paints. The principal manganese compounds which are used in the color industry are manganese white ($MnCO_3$), manganese green (manganous oxide), manganese violet (manganese metaphosphate), and manganese black (pyrolusite). Wad is used as a constituent of umber paint. Manganese hydroxide and sulphate are used as colors for porcelain and also for the dyeing of textiles. Manganese chloride is used for dyeing cotton to manganese brown or bronze and also in calico printing.

Fertilizer.- Manganese plays a very important part in plant and animal nutrition. The use of manganese in agriculture has been increased in the last few years as a result of some practical demonstrations with truck crops in southern Florida. Most soils contain sufficient manganese for profitable crop production, but in certain sections where manganese is rare in rocks and soils serious difficulties are experienced. The addition of small quantities of manganese sulphate rendered such land productive to crops heretofore considered failures.

Miscellaneous.- Manganese ore is used in the manufacture of iodine and chlorine, but the expansion of the electrolytic process for the production of chlorine has caused a decline in the demand for manganese dioxide for this purpose. Various salts of manganese are used in disinfectants, deodorizers, sterilizing agents, flotation agents, fluxes, medicinal reagents, photographic reagents, and for impregnating leather. The manufacture of the manganates and the permanganates for use as germicides and deodorizers is now an important branch of the chemical industry. The permanganates are also used for the preservation of timber, bleaching of textile fabrics, and as an oxidizing agent in the manufacture of organic compounds.

MARKETING AND PRICE

Of the major steel-producing countries, only Russia has within its national boundaries sufficient manganese ore to supply its needs; all other countries must depend upon imports for their requirements. Consequently, manganese ore is an important strategical commodity in world trade. During the war and immediately after there was a threatened shortage in the supplies of manganese ore due to the restrictions on shipping and later to revolution and internal disturbances in Russia. High prices prevailed during this period. Later, the establishment of more settled conditions in Russia and the advent of new low-cost producers resulted in increasing the supply so that there is now in normal times an oversupply, and sufficient reserves are in sight to insure consumptive industry of an assured supply. Prices declined with the increasing supply.

While there is some small trade in the carbonates and silicates, the principal ores marketed are the oxides. The character of the oxide determines the purpose for which it is used - whether metallurgical or chemical. Metallurgical ore should have a high manganese content (from 45 to 50 percent) and a low content of silica, oxygen, and phosphorus; it may be high in lime. Chemical ores should be high in oxygen, may contain considerable silica and phosphorus, but be low in lime.

Prices of manganese ore used in the metallurgical industry are quoted on a per unit basis, the unit being 1 percent of a long ton, or 22.4 pounds of metallic manganese. Prices of manganese ore imported into the United States are quoted by the trade journals according to grade and country of origin. According to the Engineering and Mining Journal the trend of prices for imported metallurgical-grade ore was as follows during 1932; the prices are given per long ton unit of manganese c.i.f. North Atlantic ports, exclusive of duty: The trend of prices of ores from all sources was downward during the year. Brazilian ore containing from 46 to 48 percent of manganese was quoted at 23 cents at the beginning of the year but finished the year at 18 cents. Chilean ore containing a minimum of 47 percent of manganese varied from 29 cents at the beginning of the year to 20 cents at the end of the year. The majority of the Chilean quotations were nominal. Indian ore containing 25 to 26 percent of manganese varied from 25 to 26 cents to 20 and 21 cents. Caucasian ore containing from 52 to 55 percent of manganese dropped from 26 cents to 22 cents. South African 52 to 54 percent grade declined from 23 and 25 cents to 20 and 21 cents The 50 and 52 percent grade was quoted from 23 to 25 cents at the beginning of the year and at 20 and 21 cents at the end of the year. The 44 to 46 percent ore dropped from 21½ cents in January to 18 or 19 cents in December. The manufacturers of ferromanganese commonly select their supplies from various sources in order to get a more favorable furnace burden. Characteristic properties of certain ores are as follows:

High-manganese content	Russian washed ore
Low-phosphorus content	Brazilian or Gold Coast ore
Low-iron content...........	Russian ore
Low-silica content.	Brazilian or Gold Coast ore
Low moisture............	Indian ore
Good structure...	Indian ore

The trend of prices of metallurgical manganese ore imported into the United States from 1915 to 1932 is shown in figure 4. The prices[15] are given in dollars per long ton of 50 percent manganese ore at North Atlantic ports. From October 1922 through December 1932, two curves are shown; the upper curve is the duty-paid price and the lower curve is the price with the duty unpaid. The figures used in the following discussion are duty-paid prices, as that figure represents the cost of the American consumers. The curve shows that the peak prices occurred during the war period and in July, 1918, reached a maximum of $68.50. With the close of the war the price dropped, until in December 1919 it was given at $27.50. A reaction upward is noted in 1920, but during the slump of 1921 prices fell to $10.50. The application of the tariff of 1922, amounting to $11.20 per long ton on 50 percent ore, was immediately reflected in the price. The tariff increment, combined with an increasing tendency in the price, caused a level in excess of $30 to be reached early in 1923. The price, though declining, remained in excess of $30 until the latter part of 1928, after which a continuous decline brought the price down to $20.70 at the end of 1932.

The lower-grade manganiferous raw materials used in the metallurgical industry are known as ferruginous manganese ore and manganiferous iron ore. The ferruginous manganese ore which contains from 10 to 35 percent of manganese and the manganiferous iron ore which contains from 5 to 10 percent of manganese may be marketed on a unit basis, but as a rule the manganiferous iron ores and the lower-grade ferruginous manganese ores are sold on the total metallic content, - that is, iron plus manganese, based upon the current price of iron ore.

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15 Steel, Manganese Ore, 1915-1932: Vol. 92, No. 1, January 2, 1933, p. 105.

The chemical industry buys manganese ore on the basis of its content of dioxide, as the oxygen in the ore is the valuable constituent. The ore should be porous and moderately hard. The greater part of this class of ore comes from Russia, the Gold Coast, the United States, Java, and Porto Rico. The ore is usually sold at a price per long ton with a minimum content of peroxide specified. According to the Engineering and Mining Journal the record of prices for chemical and battery ore during 1932 was as follows: Imported ores (Brazilian, Cuban, Caucasian, Javan) containing from 80 to 85 percent of manganese dioxide were quoted from $50 to $60 throughout the year. Domestic ores containing 70 to 72 percent of manganese dioxide were quoted at $43 to $50 f.o.b. mines.

EXPORTS AND IMPORTS

The bulk of the manganese ore produced in the world is consumed in places distant from the sources of production. The large production is in outlying, less highly developed parts of the world, whereas the consumption is, of necessity, in the highly industrialized regions. Consequently, with the exception of the output of Russia and of smaller amounts in India, virtually all of the manganese ore crosses national boundaries. Furthermore, the long haul and the large tonnages used require that manganese be transported by water. Often the rail haul is the determining economic factor in the development of deposits.

The principal exporting countries are, therefore, the largest producers. Table 5 shows the world exports of manganese ore by countries from 1926 to 1930. The principal exporters during this period were India and the U.S.S.R., each contributing about one third of the world total. India led during most of this period, but in 1929 the U.S.S.R. forged ahead only to drop back again in 1930.

The distribution of exports from the U.S.S.R. for the same period is shown in table 6. The United States was the largest consumer of the ores from U.S.S.R. during the quinquennium. France took important and increasing amounts.

The Indian ores (see table 7) moved to Belgium, France, United Kingdom, and the United States. The bulk of the Brazilian ores moved to the United States, although, as shown in table 8, important quantities were shipped to France and Belgium.

During the four years (1926-1929) shown in table 9, more than 70 percent of the Gold Coast ore moved to Canada, Norway, and the United States. The manganese imported as ores into Canada and Norway reenters the market again in the form of ferro-alloys. France also has taken significant quantities of Gold Coast ores.

Table 5.- World exports of manganese ore by principal exporting countries, 1926-1930, long tons

Country	1926		1927		1928		1929		1930	
	Quantity	Percent	Quantity	Percent	Quantity	Percent	Quantity	Percent	Quantity	Percent
Brazil	314,774	15	238,005	10	356,116	16	288,685	10	189,088	8
Gold Coast	371,324	18	403,187	16	343,246	16	457,932	16	445,605	19
India	613,536	30	843,821	35	834,144	39	964,489	34	773,026	34
U.S.S.R.	622,414	31	771,129	32	499,191	23	1,020,746	36	742,292	32
Other[1]	120,000	6	170,000	7	120,000	6	125,000	4	160,000	7
Total	2,042,048	100	2,426,142	100	2,152,697	100	2,856,852	100	2,310,011	100

[1]Estimates. Does not include export of manganese bearing ores from Egypt.

Table 6.- Exports of Manganese ore from Russia, by countries, 1926-1930, long tons

Country	1926[1] Quantity	1926[1] Per-cent	1927[1] Quantity	1927[1] Per-cent	1928[1] Quantity	1928[1] Per-cent	1929 Quantity	1929 Per-cent	1930 Quantity	1930 Per-cent
Belgium	81,603	12	70,926	9	64,868	13	74,759	7	13,999	2
France	40,251	6	79,425	10	63,513	13	117,800	12	115,105	15
Germany	41,600	6	71,803	9	43,366	9	47,377	5	91,252	12
Italy	32,156	5	31,974	4	45,612	9	56,357	6	61,791	8
Netherlands	100,719	16	240,928	32	44,243	9	-	-	44,888	6
Poland	23,133	3	21,023	3	30,333	6	38,900	4	24,849	3
United Kingdom	68,746	10	25,741	3	12,966	3	64,391	6	14,558	2
United States	274,157	42	211,140	29	186,102	38	395,025	38	182,989	25
Other	-	-	7,522	1	-	-	226,137	22	192,861	27
Total	662,365	100	760,482	100	491,003	100	1,020,746	100	742,292	100

[1]Fiscal year ending September 30.

Table 7.- Exports of manganese ore from India, by countries, 1926-1930, long tons

Country	1926 Quantity	1926 Per-cent	1927 Quantity	1927 Per-cent	1928 Quantity	1928 Per-cent	1929 Quantity	1929 Per-cent	1930 Quantity	1930 Per-cent
Belgium	185,974	30	174,485	21	183,897	22	181,174	19	98,205	13
France	151,906	25	151,100	18	195,576	23	219,055	23	208,887	28
Germany	6,346	1	12,805	2	20,945	3	12,630	1	24,360	3
Italy	9,600	2	5,150	1	9,475	1	4,246	0	620	0
Marmagoa[1]	89,620	15	162,378	19	175,577	21	184,939	19	170,577	22
Netherlands	14,800	2	12,500	1	7,501	1	36,350	4	11,500	1
United Kingdom	74,750	12	211,401	24	159,227	19	264,537	28	164,895	21
United States	67,250	11	97,500	12	76,000	9	51,250	5	54,000	7
Other countries	13,290	2	16,502	2	5,946	1	10,308	1	39,982	5
Total	613,536	100	843,821	100	834,144	100	964,489	100	773,026	100

[1]Some Indian ore is exported via Marmagoa, Portuguese East Africa. Details as to the destination of this ore are not available.

Table 8.- Exports of manganese ore from Brazil, by countries, 1926-1930, long tons

Country	1926 Quantity	1926 Per-cent	1927 Quantity	1927 Per-cent	1928 Quantity	1928 Per-cent	1929 Quantity	1929 Per-cent	1930 Quantity	1930 Per-cent
Belgium	11,340	4	53,345	22	60,318	17	30,631	11	2,329	1
France	9,821	3	25,706	11	54,897	15	29,962	10	10,926	6
United States	287,573	91	158,638	67	222,727	63	228,092	79	175,666	93
Other	6,040	2	316	0	18,174	5	-	-	167	-
Total	314,774	100	238,005	100	356,116	100	288,685	100	189,088	100

Table 9.- Exports of manganese ore from the Gold Coast, by countries, 1926-1930, long tons

Country	1926		1927		1928		1929		1930	
	Quantity	Percent	Quantity	Percent	Quantity	Percent	Quantity	Percent	Quantity	Percent
Belgium	13,889	4	29,872	7	12,565	4	62,813	14	(1)	-
Canada	49,276	13	67,689	17	53,356	15	85,751	19	(1)	-
France	19,606	5	68,593	17	78,566	23	85,944	19	(1)	-
Norway	141,826	38	157,726	39	139,444	41	169,722	37	(1)	-
United Kingdom	32,635	9	619	0	3,806	1	1,523	0	(1)	-
United States	111,990	30	78,688	20	53,464	15	50,597	11	(1)	-
Other	2,102	1	-	-	2,045	1	1,582	0	(1)	-
Total	371,324	100	403,187	100	343,246	100	457,932	100	445,605	-

[1]Not available.

The United States draws largely on imports for its necessary requirements of manganese ore; the U.S.S.R., Brazil, and the Gold Coast furnish the principal amounts which in 1931 accounted for 83 percent of the total imports. Details of the imports of manganese ore into the United States from 1927 to 1931 are shown in table 10:

Table 10.- Manganese ore imported into the United States by countries, 1927-1931, long tons

Country	1927	1928	1929	1930	1931
Union of South Africa	-	5	-	-	5,002
Gold Coast	87,230	24,186	33,593	93,142	87,439
Brazil	174,026	142,300	216,535	185,048	133,927
India British	93,017	83,600	72,940	58,150	47,850
Canada	430	3,929	4,804	15,998	18,832
Chile	3,206	9,340	2,000	3,485	1,748
Cuba	8,976	3,180	2,667	2,071	3,804
Egypt	-	-	101	-	-
Germany	65	133	1,137	66	30
Italy	-	-	71	-	-
Java and Madura	1,527	1,026	1,000	1,602	1,754
U.S.S.R.	253,544	159,842	329,336	225,888	195,834
United Kingdom	46	167	85	109	6,298
Other countries	-	-	-	9	-
Total	622,067	427,708	664,269	585,568	502,518

Although the United States is the largest producer of steel, during the 5 years ending 1931 it was second in imports of manganese ore. France imported more ore during this period, but its production of steel amounted to only 14 percent of the American output. India and the U.S.S.R. are the principal suppliers of French imports. Recent reports indicate that France is developing manganese resources within the French Empire in order to become independent of foreign supplies. Details of the French imports of manganese ore are shown in table 11:

Table 11.- Manganese ore imported into France by countries, 1927-1931, long tons

Country	1927	1928	1929	1930	1931
Brazil	22,174	42,489	70,532	16,428	3,832
India British	257,077	262,877	272,512	288,512	132,891
Gold Coast	31,984	77,203	101,107	82,551	30,579
Spain	40,546	13,031	18,969	21,370	1,565
U.S.S.R.	77,508	108,667	181,132	189,021	134,816
Other countries of Asia	34,979	41,793	36,294	12,701	30,476
Other countries	184,904	180,511	107,050	93,175	139,494
Total	649,172	726,571	787,596	703,758	473,653

Great Britain draws most of its manganese ore from within the Empire, India furnishing the principal amounts. In 1931 imports of manganese ore from India amounted to nearly 80 percent of the total British imports. Table 12 shows the imports into the United Kingdom by countries of origin from 1927 to 1931.

Table 12.- Manganese ore imported into the United Kingdom by countries, 1927-1931, long tons

Country	1927	1928	1929	1920	1931
Brazil	1	8,010	-	-	-
Gold Coast	505	1,319	1,380	2,297	5,238
India	176,757	175,378	259,260	184,326	62,168
Java	1,973	1,998	1,361	1,225	1,675
Netherlands	197	583	629	413	357
Marmagoa	-	13,005	11,256	10,323	-
Spain	3,291	-	2,289	1,270	742
U.S.S.R.	13,061	1,750	10,046	8,401	1,606
Other	2,742	3,922	3,132	8,189	6,479
Total	198,527	205,965	289,353	216,444	78,265

Germany imports most of her requirements from the U.S.S.R. India furnishes important amounts, and there is some import of the ferruginous manganese ores of Egypt. The details of German imports are shown in table 13.

Table 13.- Manganese ore imported into Germany by countries, 1927-1931, long tons

Country	1927	1928	1929	1930	1931
Brazil	2,529	27,372	3,617	523	-
India British	75,431	124,594	128,485	77,116	23,016
Gold Coast	(1)	(1)	(1)	328	234
Netherlands	6,133	12,567	7,765	3,749	3,457
Egypt	46,092	29,489	35,044	17,208	11,015
Russia in Europe	(1)	(1)	(1)		
Russia in Asia	207,312	71,060	175,933	170,921	109,697
Other countries	29,567	14,669	30,421	60,233	12,378
Total	367,064	279,751	381,265	330,078	159,797

[1]Included under other countries.

India furnished the larger part of the manganese ores imported into the Belgium-Luxemburg Economic Union during 1927 to 1931. These amounts, however, were supplemented by important receipts principally from the U.S.S.R. Further details are shown in table 14:

Table 14.- Manganese ore imported into the Belgium-Luxemburg Economic Union, 1927-1931, long tons

Country	1927	1928	1929	1930	1931
Brazil	-	37,752	18,263	1,551	-
Egypt	980	-	-	(1)	-
France	-	-	-	16,108	12,797
Gold Coast	-	-	-	7,727	1,358
India British	94,857	122,977	149,868	117,198	98,848
Portuguese India	-	12,611	19,270	17,895	9,866
U.S.S.R.	75,183	68,254	87,610	64,524	96,833
Other countries	94,090	21,333	48,466	30,540	18,596
Total	265,110	262,927	323,477	255,543	238,298

[1]Included under other countries.

Imports into Norway and Canada come largely from the Gold Coast.

POLITICAL AND COMMERCIAL CONTROL

Political Control

Of the record world production of manganese ore in 1930, 86 percent originated in two political units: The British Empire (45 percent) and the U.S.S.R. (41 percent). Brazil contributed 6 percent and the remaining 8 percent came from various other countries throughout the world.

Commercial Control

The bulk of the world's output in 1930 was controlled by three groups: Russian, British, and American. The Soviet Union, through its control of the deposits at Chiaturi and Nikopol, was responsible for more than 41 percent of the world's production of manganese ore in 1930. British interests controlled about 32 percent, chiefly in India, the Union of South Africa, and Egypt. American firms controlled 20 percent by virtue of their operations in the Gold Coast and in Brazil. The Brazilian properties are controlled through a subsidiary corporation, whereas in the Gold Coast the American firms operate under lease from British fee owners.

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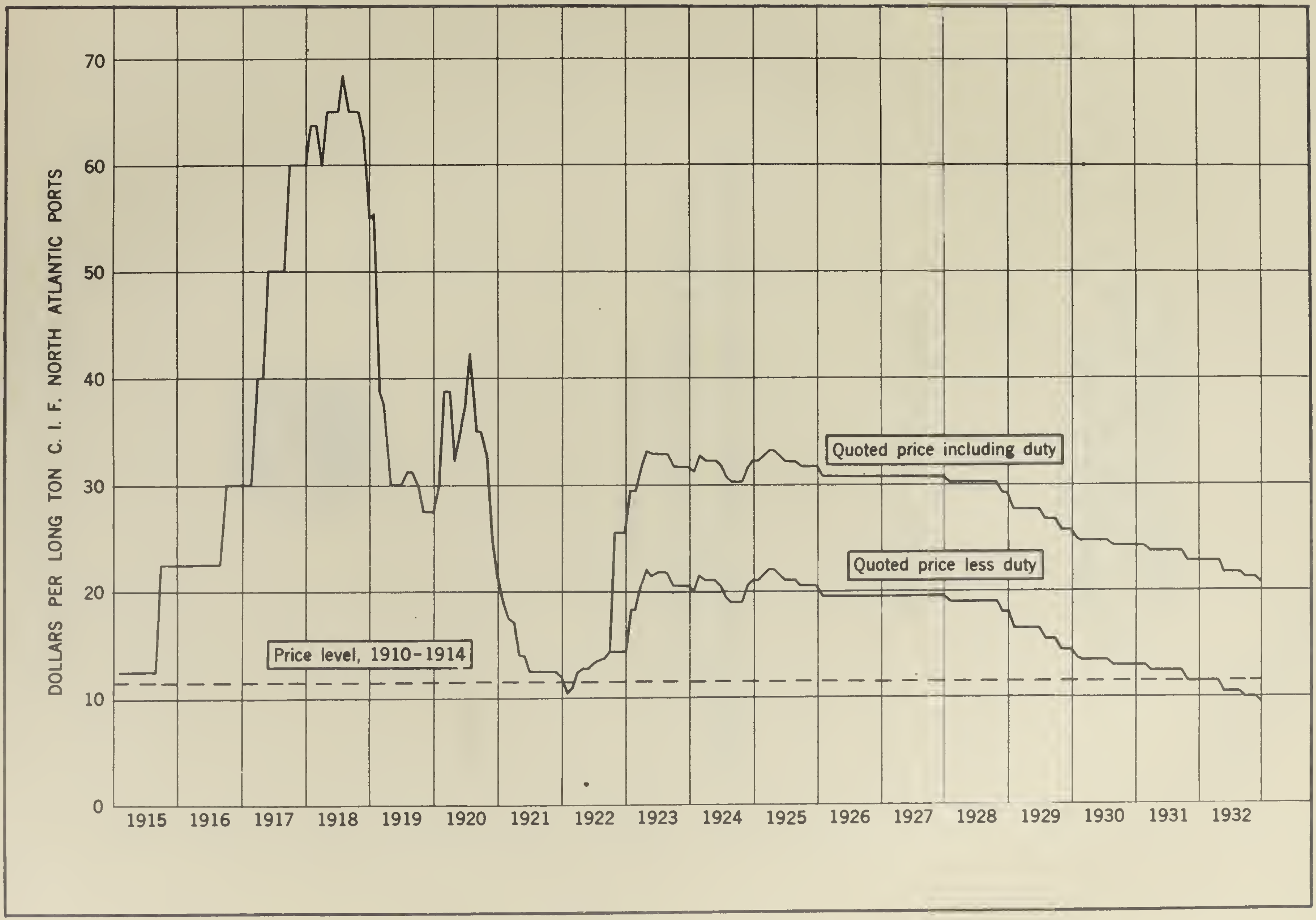

Figure 4.— Price of 50 percent manganese ore imported into the United States from 1915 to 1932.

I. C. 6730

June, 1933

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

DESIGN, EQUIPMENT, AND CONSTRUCTION COSTS OF THE DAVIS-DUNKIRK CONCENTRATOR, PRESCOTT, ARIZ.

BY

E. L. Sweeney

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

DESIGN, EQUIPMENT, AND CONSTRUCTION COSTS OF THE DAVIS-DUNKIRK CONCENTRATOR, PRESCOTT, ARIZ.[1]

By E. L. Sweeney[2]

INTRODUCTION

The mine and mill of the Davis-Dunkirk Mines, Inc., are about 15 miles by road south of Prescott, Ariz., which is the nearest railroad point for receiving mine and mill supplies and for shipping concentrates to smelters. A good trucking road reaches within 2 miles of the property, but from that point the grade is very steep. Outgoing loads are limited to 2 tons.

The topography near the mine is exceedingly rugged, and the area is slightly wooded. The climate is mild excepting in a few winter months, when the snowfall is heavy.

Electric power is secured by way of a 4 1/2-mile, high-tension line connecting with the main line of the Arizona Power Co.

The concentrator was intended to treat 60 tons or more daily of gold-silver-copper ore and make a single gold-silver-copper concentrate for shipment to a copper smelter.

DESIGN OF CONCENTRATOR

Figure 1 shows the flow sheet of the concentrator.

It was apparent from the nature of the ore that flotation was the most suitable method of concentration. Tests showed that any of the common types of flotation cells would give both a satisfactory grade of concentrate and a high percentage of recovery. The flotation cells are placed so that other arrangements such as the more usual rougher-cleaner set-up may be adopted if desired.

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6730."

2 Mining engineer, and one of the consulting engineers, U. S. Bureau of Mines.

The crushing and grinding units indicated on the flow sheet were selected because the writer's experience in several mills of 50 to 150 tons daily capacity shows that secondary crushing is uneconomical in such plants. The crusher is capable of reduction to 1 1/2-inch size, and the presence in the ball-mill feed of some slab-shaped rocks larger than this in one or two dimensions is not harmful because of the relatively great length of the mill, especially if the density of the mill discharge is properly controlled. The ball mill is driven by a V-rope drive.

Filter concentrates are discharged directly on a drying floor that is roofed over but provided with only low sides. A trapdoor discharges the concentrates from the drying floor into trucks.

No water is recovered from the tailings excepting the overflow of the thickener.

The mill is so compact that two men on each shift can operate not only the milling and concentration sections but the crusher, and in addition the mine compressor, which is installed alongside the mill.

The mill building is 30 feet wide and 55 feet long, comprising a grinding floor 20 feet long, a flotation floor 15 feet long, and a filter and thickener floor 20 feet long. The total floor area is 1,650 square feet. Little extra space was provided, excepting that a small warehouse was built at the level of the grinding floor for storage of oils, reagents, and other supplies, and ample space was also provided at that point for storage of other materials and for repairs.

As the excavation was largely in solid rock, only the thinnest practicable concrete walls were needed between the different floors. Twenty-four gage corrugated iron was used for roofing, and 26 gage for siding.

Because of the small size of the mill and the cost of freighting, timber construction was preferred to steel. The building is strongly designed, with special provision for snow load. Both coarse- and fine-ore bins are of timber construction. The grizzly and crusher are placed on top of the fine-ore bin.

Lighting is provided for by common double-sash windows and flood lights on each floor. No heating is provided for.

EQUIPMENT

The equipment used was chiefly new; nevertheless, a considerable saving was made by purchasing some equipment from a small flotation plant that had been run not more than five weeks. These items are indicated in the following tabulation which shows mill equipment costs, f.o.b. factories:

Mill equipment costs, f.o.b. factories

Items	Cost	Cost if new
1 grizzly, bar type, 3 by 6 feet by 1 1/2 inches	$ 30	$ 70
1 coarse crusher, 9 by 21 inches	1,750	1,750
1 feeder, belt type, 16 inches by 12 feet	125	250
1 ball mill, 4 1/2 by 8 feet	2,800	2,800
1 classifier, 4 1/2 by 20 feet	800	1,650
2 flotation machines, McIntosh, 10 feet	600	1,600
1 concentrate thickener, 16 feet	889	889
1 filter, drum type, 3 by 4 feet	1,060	1,950
1 Dorrco pump, 2 inches	260	260
1 tailing thickener, 30 feet	1,613	1,613
1 blower	572	572
2 pumps, Wilfley, 2 inches	200	350
3 reagent feeders	100	300
6 motors, starters, etc.	2,109	2,109
Drives	1,412	1,412
Ball load	500	500
Tools for mill operation	300	300
Total cost of equipment	15,120	18,375
Cost of equipment per ton of daily capacity (75-ton basis)	202	245

SELECTION OF MILL SITE

All of the area near the mine is too steep for an ideal mill site. Although the point selected, about 250 feet from the mine adit, is the flattest site available, the mill floors have too great differences of elevation and require more excavation than is usual on flatter sites. Other factors influencing the choice of a site were water supply, tailings disposal, and transportation, or road-building. At the place chosen, the mill water can be run by gravity into storage tanks just above the ball-mill floor. The topography is too steep for convenient tailings disposal, yet it is necessary to avoid stream pollution. An area was found a few hundred feet downstream from the mill where tailings could be banked on the hillside. In operation the cost of tailings disposal is $0.12 per ton. At the mill site selected it was possible to build branch roads to the head of the mill, to the ball-mill floor, and to the concentrate drying and storage platform.

CONSTRUCTION

The plant was completed in 82 working days, in spite of bad weather and an unusually heavy snowfall which hampered the work.

Construction costs were increased by several other factors: All sand and gravel had to be trucked from Prescott; getting heavy equipment from trucks onto the various mill floors was most difficult; the design called for unusually heavy construction; and speed was desired.

Millwrights and mechanics received $8 and common labor $5 for 8 hours.

The building and bin unit costs were as follows:

	Per square foot	Per cubic foot
Mill building only	$ 1.42	$ 0.065
Mill building and bins	1.08	.050

Details of construction costs, including everything but design and equipment, were as follows. All building material, electrical equipment, and piping were new.

Details of construction costs

Rock excavation, 770 cubic yards (in place) at $2.58 per yard	$ 2,003
Earth excavation, 600 cubic yards (in place) at $0.50 per yard	300
Concrete, 78 cubic yards at $18.00 per yard	1,404
Mill building and bins, erected	2,340
Piping	1,256
Wiring	1,150
Water tanks (two 16-foot)	470
Freight on equipment	1,663
Freight on supplies	297
Trucking from Prescott to mill site	901
Setting machinery (including moving from trucks)	3,531
Total	15,315

The total cost of the mill as erected, with some used equipment, was $30,435, or $406 per ton of daily capacity (75-ton basis).

If all new equipment had been used the cost would have been $33,690, or $449 per ton.

OPERATION

The average capacity of the plant proved to be 75 tons per 14 hours. The cost of milling to date (January 1932) has been $1.23 per ton and the percentage recovery 93.8. The average ratio of concentration has been 3.5 into 1. Concentrates shipments have averaged 3.5 percent moisture.

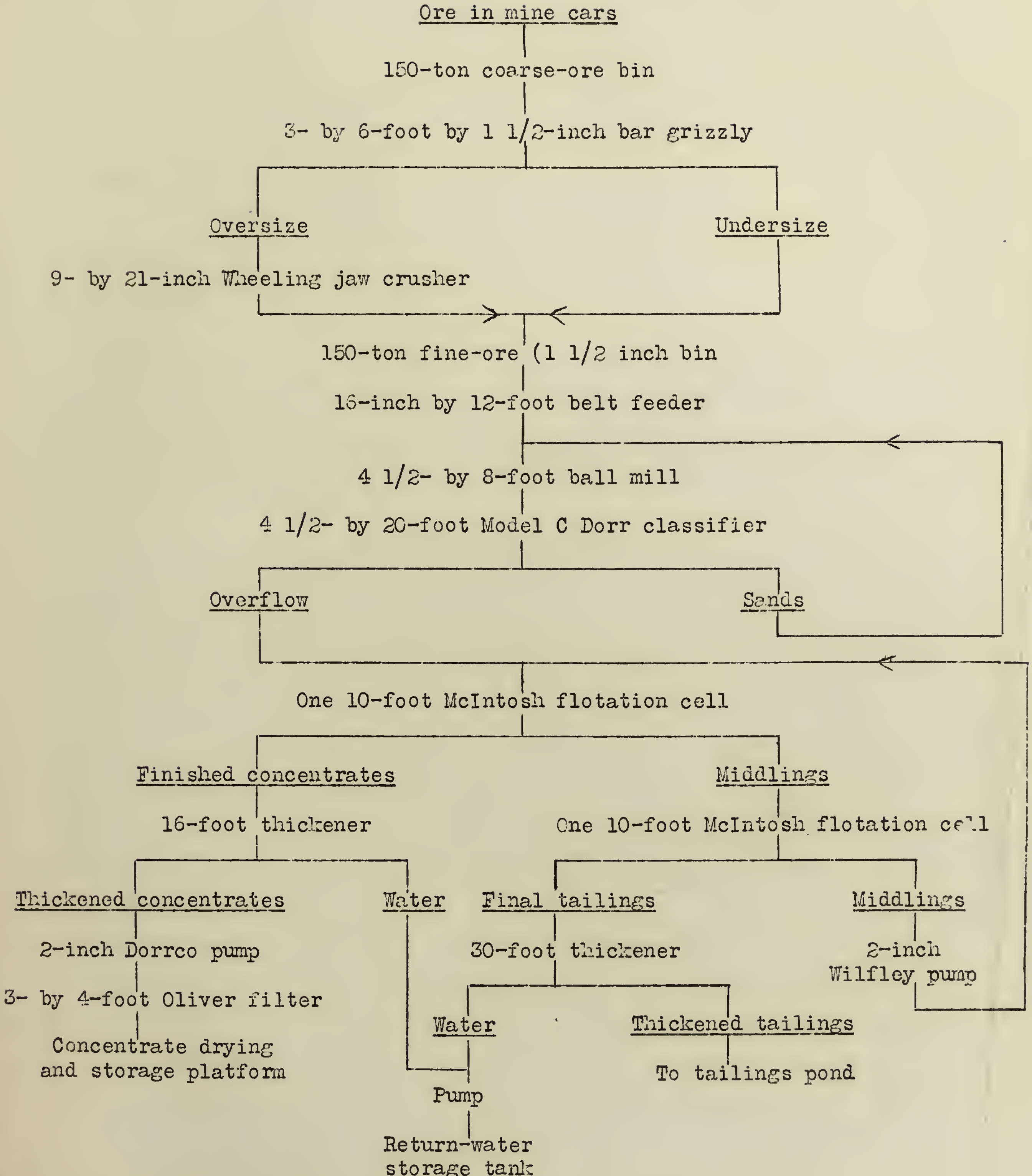

Figure 1.--Flow sheet of Davis-Dunkirk mill.

PURPOSE OF REPORT

The purpose of this report is to show what has been accomplished in one more instance where the management decided that avoidable accidents must stop and that essentially all accidents are avoidable. It describes the results of accident-prevention work and how it is being accomplished at the mines of the Pacific Coast Coal Co. Because of the relative size of the mine and the effect that its accident experience has unquestionably had on the State as a whole, this report covers primarily the safety accomplishments at the New Black Diamond mine of the Pacific Coast Co. for the period 1927-1932.

GENERAL INFORMATION

The New Black Diamond mine is in King County, Wash., in the Cedar River Valley, 6 miles east of Renton and 18 miles southeast of Seattle, on the Renton-Maple Valley highway and on the main line of the Pacific Coast Railway and the Chicago, Milwaukee, St. Paul & Pacific Railway. The mine lies in the Indian-Cedar Mountain field, which is so closely related to the Renton field that the two are usually grouped together. This joint coal area occurs in the western part of King County near the northern end of the belt of coal fields which extends in a north-south line through Lewis, Pierce, and King Counties.

The mine is opened by a 4,400-foot, 0.75-percent grade tunnel driven from Cedar River Valley at a portal altitude of 212 feet above sea level. This tunnel is timbered for its entire length; for a distance of 1,450 feet it was driven by forepoling through gravel heavily saturated with water, which at the time almost stopped operations, but later drained off. The remainder of this tunnel, although in sandstone and sandy shale, is timbered for a length of 2,950 feet. The Jones bed was cut here, and the main slope was located about 850 feet from the intersection. The tunnel averages 96 square feet in cross section inside of timbers, which are three-piece sets of 12- by 12-inch material, heavily lagged in the gravel area; it affords water-level drainage and a direct haulageway from the first level slope landing to the washing plant. The track is laid with 56-pound steel and is ballasted. The tunnel serves as the principal intake for the mine air.

The slope, which is holed to the surface above the water level at an altitude of about 600 feet, is sunk on the Jones bed to the third level. The slope is timbered 6 feet in the clear between the rail and collars, and 8 feet in the clear between the plumb posts or legs set at right angles to the roof.

The controlling grade between the top and the third level is 33° for about 210 feet between the second and third levels. The average grade is about 29°. The faults that cross the slope limit the maximum grade to 33°.

The water level, called the first level, measured on the pitch of the slope, is 673 feet from the slope portal; and the second and third levels are 1,215 and 1,728 feet, respectively, from the slope portal.

Three beds have been opened in the course of development. The beds known to be on the property, in the order of their sequence, are the Jones, the most important and the underlying bed; the Discovery, 150 feet stratigraphically above; the Cedar Mountain No. 2, 700 feet stratigraphically above the Jones, but not cut in the present mine; and the Cedar Mountain No. 1, which lies stratigraphically about 1,200 feet above the Jones.

Of particular interest in a safety discussion is the explosibility of the coal-dust from the beds being mined. Analyses[6] of the foregoing coal beds have been made. From the analyses the volatile ratio averages above 0.45, which is high, and is an index of the high explosibility of these coals; this fact is recognized by the management of the Pacific Coast Coal Co., by whom a thorough rock-dusting program is systematically carried out.

All coal is prepared by hand picking for the lump sizes, and all other sizes are washed in a Rheolaveur plant of two units. The plant reject is approximately 16 percent.

In 1930, the mine produced 320,345 tons, or 14 percent of the State total. During the 6-year period 1927-1932 the output was 1,506,543 tons, equivalent to 11.2 percent of that of the State. An average of 255 men employed underground and of 82 employed on the surface worked 2,752,928 and 892,072 man-hours, respectively. The coal mined per man-day was nearly 4.4 tons for underground men and was 3.3 tons for all employees. The State average was 3.52 tons for the period.

SURFACE PLANT AND EQUIPMENT

Coal preparation requires a more than average amount of surface equipment and personnel at the Western Washington mines; approximately 23 percent of the man-hours of exposure at the New Black Diamond mine are charged to surface operations.

The tipple, washery, and storage bins are situated about 700 feet from the mine portal and are reached by a practically level trestle. The tipple is a wooden-frame building with galvanized-iron covering; attached to it as part of the structure are large wooden bins of slow-burning material. Fire protection in the tipple and other surface buildings consists of Pyrene-type fire extinguishers placed at several points and hydrants with hose. Water tanks and several hose houses with hydrants are placed advantageously about the tipple yard, and inspection cards are displayed in these hose houses to show that the hose is in good condition. Fire drills are also held. The electric wiring is in conduit, 440 volts a.c. being used for motors and 110 volts for lighting. Pyrene extinguishers and pails of sand are well distributed about the electrical equipment. The transformers are installed in a wire-fenced area which is kept locked. Smoking is not permitted in and about the washery. Frequent systematic washing down keeps the dust at a minimum. The machinery in general is exceptionally well guarded.

6 See footnote 5b.

The surface engine house, located at the top of the Jones slope, is a wooden-frame building covered with galvanized iron.

The main hoist is electrically driven. An engineer is on duty at all times, who looks after the main ventilating fan in addition to his other duties. The hoist has a rope speed of 700 feet per minute and is driven by two 200-hp., a.c., slip-ring motors with efficient controls. In hoisting, three cars weighing 17 tons make the trip; the hoisting cable is of $1\frac{1}{4}$-inch plough steel and the car couplings are of $1\frac{1}{4}$-inch stock. The hoisting cable is inspected daily and a record is kept of its condition. The rope is removed when upon inspection it is found to be unsafe, as shown by broken wires.

A low-pressure, hand-fired boiler furnishes hot water for the washhouse. The attendant in charge of the washhouse also looks after the boiler. Steam is used at the mine only for heating and for cold-weather service around the washery.

Electric power is purchased. The power is delivered at the mine at 13,000 volts and is stepped down to 2,200 volts at a bank of transformers which consists of three 250-kv.-a. units; some power is utilized at 2,200 volts, but to give lower voltages other 2,200- to 440-volt transformers are provided. The hoist is on a separate line which is connected direct to the 13,000-volt line.

The d.c. system is all 250 volts and is generated by a motor-generator set at the plant. The a.c. side is a synchronous motor rated at 731 kw. using 2,200 volts, while the d.c. side is rated at 1,090 amperes at 250 volts. There are no haulage feed lines in the mine on the d.c. side other than the trolley line of 0000 wire. The entire track system is electric-bonded with bonds of ample capacity. Both rails are bonded in the main haulageway and one rail in the gangways.

A telephone system is efficiently maintained both on the surface and at the slope landings. The hoisting signals are clear and distinctly given. The open-type bell wires operate on 20-volt alternating current and are hung in the slope.

UNDERGROUND EQUIPMENT AND HAULAGE

Mine haulage is by rope and trolley locomotives. Sheet-iron chutes are used on the pitch and the mine cars are loaded on the gangways.

The rails on the main slope and haulage tunnel are of 56-pound steel and those on the gangways of 40-pound steel. The track gage is 36 inches and the maximum grade allowable on the gangways is 0.75 percent. Due to the easy running of the Timken-bearing mine cars, however, difficulty has been experienced in braking the trips, and the gangway grade is held, if at all possible, to 0.5 percent.

The main haulage motor for the outside haulage of coal from the Jones slope to the tipple is a 250-volt, 12-ton Westinghouse locomotive of the outside-wheel type equipped with air brakes. It makes a run of 10,200 feet per round trip, hauls 40 or 50 cars per trip, and moves from 480 to 500 cars per 8-hour day. Smaller locomotives are used on the various gangways.

Mine cars are of wood and steel construction with solid ends and without brakes. Each car has a capacity of 122 cubic feet, holds approximately 3 tons of recoverable coal, and when empty weighs 3,889 pounds. The dimensions are 13 feet 1 inch long, 4 feet $7\frac{1}{4}$ inches wide, and 4 feet 3 inches high over-all. Sixteen-inch diameter wheels are set on a 4-foot wheelbase and are equipped with Timken roller bearings using grease.

The type of car and character of loading does not permit topping, and any spillage of coal on the roadways is due to the coal's piling up on one side while being loaded from the chutes. The coal, with the exception of the gangway faces, is dry and has a tendency in some localities on the pitch to make considerable fine dust, which tends to accumulate in the pitch workings above the counter.

Pumping is relatively not a serious problem, as the water to be removed probably averages only about 250 gallons per minute throughout the year. For this purpose electrically-driven, induction, a.c. pump units are installed in separate splits of intake air. Compressed-air pumps are used for sinking or where there is likely to be an explosion hazard.

Manways and Man-Trips

The mine is provided with good manways in the slope return air course, and in addition separate fresh-air, main-intake traveling ways are provided. Manways with ladders are provided in all chutes on the pitch.

Man-trips are furnished for the travel of the men into and out of the slope, as well as out of the main water level to the surface. On the slope, safety man-cars of the Klansnic type are handled in 4-car trips and carry 10 men to the car. Regular mine cars are used on the water level behind the trolley locomotive, and no one is permitted to ride loaded trips.

UNDERGROUND MINING CONDITIONS AND METHODS

Characteristics of Jones Bed

Because most of the experience discussed in this report was developed in the Jones bed, its characteristics are given. This bed is worked for its full thickness of 7 to 8 feet. It has a strong shale roof which contains numerous coal streaks. On account of its shelly nature about 2 inches of the rock lying next to the coal comes down with the mined coal; this cannot be prevented, but is not a dangerous factor in a cap rock. The dip of the

beds averages 30°, increasing to the south and west up to about 40°; the dip permits the use of sheet-iron chutes, and the coal runs well. The bottom is banded and bony; it heaves in the pillar operations, breaking the props, but usually gives advance warning. Numerous small faults run essentially parallel to each other through the bed, but there is little or no cross faulting and the roof stands up well. The faults tend to relieve pressure on the pillars, thus making it necessary either to mine the coal or blast it off the solid. Niggerheads are found in this bed and usually occur in the lower bench or in the interval between the two benches; they are generally small, and stand almost vertical, resembling stumps of old tree trunks.

Plan of Underground Development

Levels are spaced 550 feet apart, measured from high to low rib of levels. Auxiliary slopes are used below the main levels when sinking is done. The main slope has a single track; the turnouts are made on a 100-foot radius, and the main partings hold 40 cars. During sinking operations, water is handled by means of a compressed-air pump kept within working distance of the face.

Two air courses, one on each side of the slope and parallel to it, serve as main returns and can also be used for manways; they are about 10 feet wide, the height of the bed, and are timbered with chute sets of 7-inch round timbers. The slope pillars are about 110 feet wide and are solid between the main slope and the airways. The main airways are holed to the surface near the slope mouth, and the mine fan is located at the mouth of one of them.

Method of Mining

The method of mining is known as the "compartment chute-and-pillar system" and is described in detail in various publications.[7]

Whether the coal is undercut, overcut, or sheared depends largely on the location in the mine; at the present time cutting is done by pick. In Washington mining practice, blasting off the solid in advance work is broadly construed to mean blasting where there is only one free face; and in pillar work, where the solid coal intervening between any free face and the back of the hole is in excess of 6 feet. The tendency is to depend upon the judgment of the supervisor as to whether or not a shot is "dependent"; this leaves room for an entirely too liberal interpretation, and it would be better to have a fairly rigid rule and adhere to it. Next to open lights, blasting is the most frequent cause of Washington's mine explosions[8] and accounts for 30 percent of the total when open lights are included. Mine operators should be

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7 See footnote 5a.
See footnote 5b.
Ash, S. H., Prevention of Accidents from Falls of Roof: Proc. Rocky Mountain Coal Min. Inst., 1930, pp. 39-43.

8 See footnote 5c.

encouraged to develop systems of mining whereby the coal can be economically and properly cut or sheared with safe equipment, rather than prohibited by inflexible regulations from introducing efficient systems to promote safer mining and to meet competition. To prepare the faces for safe blasting by pick mining on heavy pitches is highly dangerous and cannot be done in many instances because of numerous dangerous factors, including that of falling and sliding material; solid shooting is eventually resorted to, with its innumerable dangers and inefficiencies.

In the past, air-driven machines of the post-puncher type were largely used[9] at some mines of the Pacific Coast Coal Co., including the New Black Diamond mine, as well as in other regions having pitching coal beds, but they were not satisfactory or economical. When this equipment was discontinued at the New Black Diamond, it practically disappeared in that mining region. Eighteen Sullivan post-punchers were in use at one time in this mine. Where these machines were installed two men were employed to give instructions at the face in the use and care of the machines. The machines were compressed-air driven, and as designed at that time required about 250 cubic feet of free air per minute; a new machine requires only 150 to 200 cubic feet per minute.

On the water level, longwall machines were tried, and, so far as the machines were concerned, they worked satisfactorily. Unsatisfactory layout of the longwall face and difficulties with an unsatisfactory conveyor installation led to the abandonment of this type of chain machine.

Recent developments in the Roslyn field have clearly demonstrated that shortwall chain machines with present-day conveyors can be worked on pitches steeper than 45° more successfully than has been approximated by hand methods. When using conveyors, the pitch is an advantage instead of a disadvantage.

Timbering

Systematic timbering rules are strictly enforced. The pitch workings are timbered systematically with props and stringers in all parts of the mine; the gangways are timbered with sets placed on 6 to 8-foot centers. In the pillars, single-stick timbering only is required, props being set 4 feet apart, with 6-foot lagging for cap pieces. The props are of split timber, 8 feet long, and average 7 inches in diameter.

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9 Ash, S. H., Systems of Coal Mining in Washington: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 833-873.

[illegible]

Timber and Material Distribution

The method of timber distribution employed is one of the major causes of the excellent performance of the pitch miners in this mine as compared with the average for the district. The fact that plenty of material with which to work is kept readily available at the face is in itself conducive to safer working conditions at the face, where most accidents occur. During the day, or on idle days, timber is loaded in mine cars outside and brought into the mine; it is then taken to the working places either on the afternoon shift, if working full time, or at any rate in such manner as not to obstruct the face worker. This practice also applies to the handling of sheet iron, pipe, and other supplies.

The distribution system, in addition to providing plenty of timber and supplies, has materially lowered the cost of such items as compared with the old method in which men "packed" all timber and supplies. Timber "packers" still distribute some of the material on the pitch, but the bulk of the material used in advancing the chutes and in mining the pillars is moved direct from the gangway to the face of each chute or pillar.

The timbers used in chute construction, and the sheet iron and air pipes are heavy and bulky. To facilitate handling of these supplies a track is carried in the manway side of every chute. This is made of 2- by 4-inch lumber nailed over 1- by 6-inch boards to form two troughs which are used for rails. These are nailed ladder fashion over 2- by 4-inch rungs, which serve as ties as well as for the ladder. The car is 24-inch gage and is equipped with flat wheels to run in the trough rails. A truck is placed in each chute manway, and a 3/8-inch cable is attached. Extra cable for advance is made into a roll and hung on the gangway. A bull wheel is attached to the last wing near the face. Two small portable electric hoists having a drum to which the cables can be attached or removed, are used to hoist the material up the chute; the hoists are small and easily moved from chute to chute on the gangway by means of a trolley locomotive.

During the shift while the miners are at work the fire boss or shot firer takes an order for their supplies. This order is handed to the timber hoistman, if one is employed, who attends to the delivery of this material on the afternoon shift, when no coal is handled except to load all empty cars then in the mine. An extra supply of timber and pitch supplies is kept on the gangway to prevent delay on the part of the timber distributors in getting the timber to the faces during their shift. The principal feature of the distribution method consists of the track and truck in every chute, a method that eliminates the passage of timber through the crosscuts. The timber is hoisted to the last wing placed by the miner and is then carried to the face by the miner, on the basis of his contract; for this reason it is advisable for the miner to keep the wing and bull wheel as near as possible to the face. The wings are advanced each time the chute advances 20 feet.

EXPLOSIVES

Permissible explosive is used exclusively at the New Black Diamond mine for the coal, and gelatin dynamites are used in rock work. Data covering normal operations and development indicate the following averages for explosives consumption per ton of coal: Tamping bags, 0.73; blasting caps, 0.314; fuse, 1.985 feet; dynamite, 0.019 pound; and permissible explosive, 0.422 pound.

Explosives are distributed in an excellent manner and detonators are handled separately by certified shot firers who inspect and blast all shots at regular shooting times when the men are in the mine.

An examination of the working place is made by the shot firer immediately before he lights any shots. The method of lighting the fuse by means of touch paper[10] or punk is probably peculiar to this region. This practice puts much responsibility in the hands of one person; hence, the most rigid discipline is practiced at this mine relative to testing for gas. That the system is carefully used is attested by the fact that there has never been an explosion at this mine.

A record is kept of missed shots, of which there are few; when they do occur it is obviously due to one or more of the numerous difficulties likely to be met in the use of fuse. In such instances the place is fenced off and no one is permitted to return until 8 hours have elapsed. No attempt is made to recover the explosive, and another hole is drilled to one side; care is exercised in cleaning up the broken coal to obtain the unexploded explosive.

There is much that can be said in favor of electric blasting and little against it, though it is by no means "foolproof." However, under existing mining methods on the steep pitches of western Washington, it has been found in some instances to be exceedingly dangerous to fire shots singly by electric blasting methods, following more than one to two shots. It is decidedly dangerous in such instances to return in succession to the face after each shot because of falling material and impaired ventilation.

Moreover, because no acceptable and adaptable cutting machine is now available unless post punchers are used, and they are by no means satisfactory, with the present mining methods which require working up the pitch, to blast a round of shots simultaneously with a face not properly cut or sheared would be very likely to cause blown-out shots. Without exception, Washington coal-dusts in suspension are dangerous as to possible ignition or as to propagation of an explosion. Moreover, there is no permissible blasting unit for multiple shot firing. Unquestionably, in many instances permissible mining machines could be utilized safely and efficiently on moderately steep pitches to avoid solid

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10 Parker, D. J., Touch Paper: Inf. Circ. 6067, Bureau of Mines, 1928, 2 pp.

shooting; there is an honest difference of opinion as to the economic phases of methods of doing this, and there has been no practical demonstration of the feasibility of using such machines on the steeper pitches. Mines have been closed in western Washington because the coal could not be economically prepared for safe blasting practice, "electrical" or "fuse and cap", under gassy and dusty conditions. The blasting of shots when all men are out of the mine does not offer an entirely satisfactory solution for the reason that running ground is prevalent in some instances, though fortunately they are exceptional. Timbering in most cases must be extended soon after blasting; under some circumstances if it were not done the place would be lost. Fine material from the roof, slides from the floor, slabs from the roof, sloughing ribs, and niggerheads as well as other conditions make blasting an unusually dangerous practice in many Washington mines. Unquestionably, the factors involved in one way or another from blasting in this State as a whole offer the greatest disaster hazard to Washington miners.

The field is open for an adaptable permissible coal-cutting machine and permissible multiple-shot blasting units by which the coal can be properly undercut and the shots blasted simultaneously and more safely. Electric blasting from power circuits is highly dangerous on pitching high-volatile ratio coal beds unless the current control is such as not to ignite gas or the dangerous dust cloud inevitably set up by friable coals running down the pitch. However, electric blasting offers the safest way to prevent explosives accidents; and electric blasting, if properly done, offers the best safeguard against gas or dust ignitions from blasting practice; but unless due consideration and care are exercised, power blasting can prove to be far more dangerous than the use of fuse and detonators if an electric arc is present in a dangerous dust cloud, possibly containing varying amounts of methane. About 41 percent of the State's tonnage is obtained from mines using electric blasting by single-shot blasting units, - a method applicable where steep-pitch conditions are not found.

Drilling

All drilling in the mine is done by compressed-air drills supplied by the company, for which the miners furnish the steel. Each pair of miners is furnished with a drill. Drilling time is cut down considerably by this method in these particular beds, and the holes are better placed. Hand drilling was a slow method and it was difficult to secure the proper number and position of holes; one half day was the average time needed in hand drilling to place a round which now requires about 30 minutes. The use of compressed-air drills is one of the main causes contributing to better than normal or average performance of the miners in this mine. The holes are more likely to be properly placed so far as dependent shots are concerned and are drilled deep enough to allow the explosive to go to the back of the hole.

The only fatal accident that has occurred at this mine due to blasting was on December 7, 1925, when a premature explosion killed two men; this explosion was caused by a haulage motor's accidentally shorting the lead wires of primers made up for electric blasting.

VENTILATION

The New Black Diamond mine is classed as gassy and is worked exclusively with closed lights. It is ventilated by a continuously exhausting centrifugal fan. This is a Sirocco 14, double-inlet, reversible type and the fan duct is equipped with explosion doors. The fan is located on the surface near the slope portal and is connected with the main return.

Any falls in the main return and counters are cleaned up, and these passageways are kept in good condition because they are used for manways.

Permanent overcasts are in solid rock. Chute stoppings are of wood, and line brattices of wood and canvas convey the air from the last crosscut to the faces in the chutes.

The old workings are ventilated when not sealed off. The mine is kept free of standing gas, and no explosive gas is allowed to exist in the mine workings. The mine is examined for gas not longer than 2 hours prior to the time that the miner reaches the working place. Two fire bosses do this work and make their report before the men enter the mine. A record of such inspection is kept and the only time gas appears to be found is when a brattice gets too far behind. Unlocked flame safety lamps of the Wolf and Koehler type are used.

Permissible electric cap lamps of the Edison type are used exclusively for lighting by the workers. There have been no accidents from methane or from ignition of methane at this mine.

Although classed as a gassy mine the New Black Diamond is not as gassy as most deep mines in Washington. Analyses by the United States Bureau of Mines show about 2,500 cubic feet of methane per 24 hours being discharged at the main fan.

The U.C.C. methane indicator is frequently used to check the return air and ventilation maintained to keep the methane content below 0.2 percent under any conditions. Systematic sampling of the mine air is done and the samples are analyzed either by the Bureau of Mines or a thoroughly equipped laboratory maintained by the company.

ROCK-DUSTING

Moisture conditions throughout the mine as a whole, coupled with the explosibility of the fine coal-dust, justify serious consideration and effective methods of controlling what would be a dangerous explosion hazard. To protect against this danger effectively the mine is adequately rock-dusted in all accessible parts whether in a dry or even wet condition. Limestone-dust specified as satisfactory by the Bureau of Mines is used for this purpose. Rock-dust is applied and extended by hand as well as with a high-pressure M.S.A. rock-dusting machine. Systematic sampling is done to insure adequate rock-dusting.

ACCIDENT PREVENTION

First-Aid and Mine-Rescue Equipment

The mining company keeps first-aid material, including bandages, stretchers, splints, and blankets, at convenient points in and about the mine and has a doctor on duty at the New Black Diamond. An emergency hospital with attendants and ambulance service is maintained.

The company has a first-class rescue station at the mine equipped with 6 sets of Gibbs breathing apparatus, an oxygen pump, a large supply of regenerating material, oxygen, and other accessories; this equipment is kept in condition by a regular attendant and regular tests are made and reported by him. Six All-Service gas masks with adequate supplies are also kept at the station. Considerable numbers of men employed at the mines are trained in mine rescue methods and several are kept trained for emergencies. All persons employed underground provide themselves with self-rescuers, which are systematically handled and checked daily.

Welfare Work

The welfare work includes the maintenance of a mutual group insurance fund through a mine association to which all workmen belong and from which benefits are paid to the sick as well as the injured. The association is known as the Mutual Benefit Association, and is governed by rules promulgated and supervised through a committee of the mine council or workmen's organization. The monthly dues are $1.50.

Under the plan, if an employee is incapacitated through illness or injury he receives $1 per day for 150 days during any one yearly period, with the exception that for periods up to 30 days the first 7 days are exempt; but beyond this time the waiting period is retroactive.

In event of a fatal injury the beneficiary of the deceased workman receives an immediate cash settlement of $500; there has been only one fatality in 4 years.

This plan in no manner interferes with or shares in any industrial insurance system or any beneficial order to which the workmen can or may belong.

Unquestionably, such a plan tends to keep the workmen mutually concerned about the status of an injured employee, as well as the nature and cause of his injuries. It also engenders a sense of responsibility and desire to eliminate accidents, besides calling attention to those who for some cause or other are more or less frequently incapacitated.

Safety Organization

By virtue of their contacts and experience persons engaged in hazardous industries unquestionably should be competent to know the causes of accidents and also should be well aware of the misery that follows in their wake. There is probably no single endeavor that has brought to the minds of the individual how serious an injury can be and how necessary and possible it is to prevent occurrence of injuries than a well-grounded training in first aid. Experience indicates that plants keeping up first-aid training are eventually the plants that prove that accidents can be prevented. The principle of accident prevention by safe practices eventually wins when a procedure or accident-prevention policy is followed which is both educational and administrative.

At the mines of the Pacific Coast Coal Co. the educational program started in earnest about the close of the year 1928. The management then decided that every employee should be trained in first aid and that this status should be maintained; that the supervising force from the top down, including the medical staff, would be jointly responsible for accidents to men in their charge; that preventive measures should be enforced by seeing that safeguards were installed and efficiently kept; that each new man should be physically and mentally fit, provided with safety equipment that he must use, and should be taught the safe method of doing his work; and that men who were careless, unfit, or injured repeatedly, should be dismissed or shifted to work that they could perform safely.

There is probably no group of miners in the United States that are more familiar with recovery work involved in mine explosions, mine fires, and disastrous cave-ins than are the workmen in the Washington coal mines. As a fundamental training in the safe procedure to follow not only to prevent such disasters but also to handle them if they do occur, the supervising force and a large number of workmen at the mines of the company have been thoroughly grounded in (and from time to time review) the courses offered by the United States Bureau of Mines in advanced mine rescue and accident prevention in bituminous-coal mines.

The administrative program consists of an investigation of all accidents and a review of them by the management; a statistical analysis of current accidents; and the keeping of employment records showing entering and leaving service data, which are in substance histories of insurance carried by the

employee, previous employment history, previous accident record, accident and sickness history, and reasons for leaving service. Compulsory physical examination and medical service, reporting of accidents, following up compensation and medical-aid matters, the doctor's part on the job, and the complete history of claims are matters not only discussed with the supervisors but also reviewed in some part at monthly safety meetings of all employees. Current information circulars and pertinent literature of the Bureau of Mines are reviewed.

A safety committee required by State law[11] makes a bimonthly inspection of conditions affecting the safety and welfare of the men in and about the mine. This committee is composed of representatives of the employees chosen by the employees themselves, and the superintendent or some other official representing the company. This committee investigates and reports all serious or fatal accidents, the latter to the State Mine Inspector. The method of reporting accidents is prescribed by law, in that any person injured must report the nature and time of his injury to his immediate supervisor at the time of the accident.

A most effective method to keep supervisors on the alert as to what is expected of them is to let them know that the management is thoroughly informed of injuries which occur and that they expect accidents to be eliminated. To attain this end a monthly letter is addressed to each superintendent and a copy is given to each supervisor; this letter consists of a description of each accident that occurred during the month and places the responsibility. A statistical table shows a summary of accidents and severity for the month, year, and corresponding period of the year before, together with the name of the supervisor and the accident record of the district in his charge.

In the matter of protective clothing the Pacific Coast Coal Co. has set the pace by being the first to adopt compulsory head protection. A review of the number of eye injuries plainly indicates that the adoption of goggles will show results in the future. A unique system known as the "daily broadcast" is in effect; the men on congregating at the mine portal are told by means of a loud speaker connected with a microphone at the mine office of various phases of the safety of the mine, and their attention is called to current accidents and means of preventing their recurrence.

ACCIDENT EXPERIENCE

Because of the thoroughness of the statistics kept by the large individual mines and the unified compensation system throughout the State, a complete study of mine-accident occurrence is possible and is of value and importance in pointing the way to steps that should be taken to eliminate accidents.

11 Ash, S. H., Safety Committees in the Coal Mines of the State of Washington: Inf. Circ. 6283, Bureau of Mines, 1931, 9 pp.

The last fatal accident occurred at the New Black Diamond mine on January 21, 1929. On that date a miner received injuries from which he died when the "wind" caused by a cave in a pillar hurled him violently against the rib of a crosscut into which he and his partner had gone.

Since that time and up to March 1, 1933, 1,116,262 net tons of coal have been produced and 2,435,616 man-hours of labor performed; this is an outstanding record in the coal-mining history of the steep pitches of Washington. In recognition of safety achievements, the Joseph A. Holmes Safety Association on March 6, 1933, awarded a certificate of honor to this mine.

To show the effect and trends of accidents and their relation to accident rates and industrial insurance costs, a number of tables and graphs were prepared, but lack of funds prevents their being shown. The scale of time losses for weighting deaths and permanent injuries to show the severity of accidents is the same as that used by the National Safety Competition.[12] Temporary disabilities include hernias and are weighted according to the actual number of calendar days of disability, including Sundays and holidays. For industrial insurance purposes Washington has an absolute 3-day waiting period.

The first outstanding fact noticed in an accident-prevention campaign is the elimination of noncompensable accidents and fatalities. The matter of eliminating permanent partial disabilities is a problem affected not only by accident occurrence but also by the increasing tendency in all jurisdictions throughout the country to consider accidents as belonging to this class to an increasing degree.

Accidents are classified according to the supervising arrangements. A miner, for example, may be injured on his way to and from work, and obviously, whether the accident should be charged to mining, ventilation, haulage, or other department depends on the place and cause of the injury. Again, a man may be removed from his regular class of work to another. Injuries from blasting operations will be found under ventilation for the reason that shot firers and fire bosses are responsible for adequate ventilation as well as shot firing, and safety practices as they relate to blasting are an important part of their duties.

One table showed the number of injuries, time lost, accident rates and percentages by causes for all injuries during the period 1929-1932 at the New Black Diamond mine. Outstanding points of interest under "Preparation and loading" (13 accidents and 450 days) are the accidents from railroad cars (4 accidents and 168 days); under "Haulage and hoisting" (32 accidents and 8,978 days) those from getting off trips while moving (5 accidents and 452 days) and being struck by coal during loading operations underground (2 accidents and 1,882 days); under "Ventilation and light" (9 accidents and 7,675

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12 Adams, W. W., The National Safety Competition of 1929: Report of Investigations 3019, Bureau of Mines, 1930, p. 6.

days) those from being struck by falling material following blasting (5 accidents and 3,138 days); under "Mining" (222 accidents and 38,934 days), being struck in the eyes by flying coal from the pick (21 accidents and 264 days) and by falling roof while in the act of setting timber (30 accidents and 12,780 days). This last cause indicates that proper inspection and testing of the roof is lacking; the excuse of "just getting ready to set a timber" may be sometimes used by the workmen to give a reason. Supervisors are constantly warning the men and discipline is now taking an added significance. In general, slipping and falling are important, as they caused 42 accidents and 6,919 days of lost time. In the 4-year period, underground work accounted for 273 accidents and 55,772 lost days, and surface activities for 24 accidents and 626 lost days.

Another table showed the number of injuries and time lost according to labor classification, occupation, and percentage of man-hours of exposure for the labor classification, the features being given in the preceding paragraph. The variable working time makes it of value to consider the man-hours of exposure in employment rather than the average number of men employed in a certain occupation.

The accident frequency and severity for underground employees was 147.6 and 30.2, and for surface employees was 43.4 and 1.1, respectively, for the period 1929-1932. Miners have the greatest frequency rate (192.1) and the labor classification "Ventilation and light" has the greatest severity rate (98.4) as a result of injuries to miners and others caused by being struck by falling material following blasting.

The graphs prepared but not shown here reveal plainly the beneficial results of accident-prevention work. Even with the inclusion of the explosion[13] of 1930 whereby 17 men were killed the accident rate for the State shows a trend downward and the New Black Diamond mine holds an enviable position. The severity for the State as a whole is due to the severity of permanent total disability injuries.

Another table showed the number of injuries by the nature of injury and part of body injured for the period 1929-1932 at the New Black Diamond mine. The result of physical examinations is convincingly shown by the development of only two hernias in this period. The decrease in the number of eye injuries shows the importance of the use of goggles, now required. Two eyes were destroyed and 30 eyes were injured in the period. As a result of accident-prevention work fractures were reduced to 53, including 1 fatal and 2 permanent partial disabilities, entailing a loss of 29,276 days. There were 7 infections in 1929, none in 1930 and 1931, and 3 in 1932. Sprains (ankle and knee) and strains (back, side, and groin) continue, yet are fewer in number; there were 5 sprains in 1929 and 2 in 1932, and 12 strains in 1929 and 2 in 1932.

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13 See footnote 5c.

COST OF ACCIDENTS

Compensation benefits are paid for injuries received, and medical care is given at the New Black Diamond mine under regulations governing all mines of the State as prescribed by State law and administered by the State Department of Labor and Industries. It is beyond the scope of this report to discuss the compensation system. The subject as it relates to the mining industry has been discussed in detail in other publications[14, 15] of the Bureau of Mines.

Briefly, Washington's industrial insurance system operates on an exclusive basis. Compensation rates have been based on claim losses per $100 of pay roll, while medical cost rates are based on claim losses per 8-hour man-day of exposure. There is no overhead charge for compensation insurance, as this cost is cared for by appropriation of the State and paid by the general taxpayer. This charge has averaged approximately 7 percent[16] of the premiums paid. The segregated medical-aid fund and no direct cost of overhead or administration to the industry are unique features in the Washington law. The industrial insurance costs as given in this report therefore represent both the pure premium and net total cost to the industry. Medical aid is unlimited and is for all injuries whether compensable or noncompensable; one half the cost is borne by the employee. The man-day rate is fixed for all mines alike, but can be handled by the State or by the individual plant. In practice, each industry functions essentially as a mutual insurance association responsible for its own losses and under State auspices. Rates are based on the class experience and their application is largely controlled by the industry itself. It has been the policy of this industry not to carry any large reserve to cover the cost of catastrophes but to meet such costs, after conference with all concerned, by graduated assessments on the tonnage basis and distributed to each operation alike. Merit credits are given on an equal percentage basis for all plants, irrespective of size of pay roll for accident-prevention work as reflected through surplus over claim losses. That the system has met the catastrophe losses of the class successfully, and there have been several, is testified to by the fact that at the close of 1932 the class had a reserve of approximately $100,000 after paying claim losses for compensation alone of about $3,500,000 during a period of 20 years.

The segregated medical-aid fund on a man-day basis clearly indicates whether costs are increasing or decreasing. It also places the charges equitably where they belong, permits a careful scrutiny of medical costs, and eliminates an indicated weakness of the indivisible premium of "robbing Peter to pay Paul." Further, it is consistent with accident rates, is reconcilable with man-hours of exposure, is not affected by fluctuating wage scales, controls effectively the anticipated earned premiums, and

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14 See footnote 4.

15 Ash, S. H., Accident Experience and Cost of Accidents in Washington Metal Mines and Quarries: Tech. Paper 514, Bureau of Mines, 1932, 35 pp.

16 See footnote 15.

eliminates the necessity of assessments to compensate diminished premium income. Where medical fees are fixed, a just and equitable charge is applied to all plants alike. The same reasoning applied to the compensation item will go a long way toward eliminating much of the misunderstanding incident to assessment on the pay-roll basis and will not penalize the operator who pays higher wages for the same man-hours of exposure under the same conditions and same class of work. It will tend to reconcile accident rates with accident costs and will permit the indicated severity of accidents to lead the way to investigation and analysis of the costs and elimination of various classes of injuries.

The unique feature insofar as United States compensation systems are concerned was interpreted into law by the State legislature of Washington in March, 1933. By this legislation, contribution to the accident fund (compensation item) was changed from the pay-roll to the man-hour basis. (H.B. 435.)

During the period 1927-1931 the New Black Diamond mine produced 11 percent of the State total coal tonnage and worked 11.6 percent of the total time for all State mines. The Black Diamond had 3 fatalities and produced 436,456 tons per fatality, whereas the respective figures for the State were 73 killed and 162,616 tons produced per fatality. The mine's showing for the period 1929-1932 is much better and the State's is about the same. The net compensation cost to the Black Diamond from 1927 through 1931 was $111,167, or $22,233 a year. This included medical cost and special assessments, also merit rating credits of the last three years which totaled $9,591.

Calculated trends as shown in the graphs prepared but not reproduced here definitely indicate that safety work pays as it relates to accident costs. The variance of the three bases mentioned shows the advisability of their consideration. In the following listings the trends are noted as "upward" or "downward" for the periods indicated:

Trends of industrial insurance costs in Washington and
New Black Diamond mine

State of Washington, 1927-1931		New Black Diamond mine 1927-1931	New Black Diamond mine 1928-1931
Pay-roll basis:			
Compensation	Upward	Downward	Downward
Medical	Upward	Downward	Downward
Industrial insurance	Upward	Downward	Downward
Man-hour basis:			
Compensation	Upward	Upward	Downward
Medical	Downward	Downward	Downward
Industrial insurance	Upward	Upward	Downward
Tonnage basis:			
Compensation	Downward	Downward	Downward
Medical	Downward	Downward	Downward
Industrial insurance	Downward	Downward	Downward

The difference in the effect of the pay-roll and man-day basis is at once apparent in noting the medical cost trend which is on a man-day basis for the class as a whole and therefore is the same both for the mine and for the State. It will be observed that the medical costs per man-day of exposure are downward, which also reflects Washington's improvement in this item of industrial insurance cost. This probably is due to two things: (1) The system of more or less general first-aid training (76 percent of the State's mine employees are so trained); and (2) the coordinated effort of the employees, workmen, and the State through the medical system in effect. Referring to the pay-roll basis for medical trends, it will be observed that the State trend was upward while the New Black Diamond was downward. This was due entirely to a decreasing man-day wage for the State and an increasing one for New Black Diamond. The increasing average man-day wage for New Black Diamond is accounted for by the fact that efficiency in mine operation has increased through the reduction of forces on dead work, development, maintenance, and efficient planning; this increased the production per man-day from 2.52 tons in 1927 to 3.41 tons in 1932, whereas the whole State remained practically stationary for that period -- 3.35 tons in 1927 and 3.46 tons in 1932. The reduction of force on day work other than breaking coal obviously raises the production per man-day, as there are more men in proportion directly producing coal. For this reason the production per man-day could increase and there still not be an increase in the production per miner-day; however, this has also increased on account of the working of more productive areas. Being on a contract basis, the average wage level of the miners therefore increased; this fact, coupled with idle time and the steady employment of a nominal force of supervisors on a wage level slightly higher than day labor, has tended to raise the wage level in this mine.

Mechanization has played no part in increasing the tonnage per man-day at this mine, and for this reason the tonnage basis of comparison is significant as to accident-prevention work and efficiency with the same mining methods.

Many mines measure the results of safety work by the reduction in the number of accidents rather than by reduction (1) in units of production per accident; (2) number of man-hours of exposure per accident (frequency and severity); and (3) cost of injuries per $100 of pay roll, per man-hour of exposure, and per unit of production.

The cost per man-hour of exposure is the only basis that does not vary except with variation in the injury rates and cost schedules awarded for injuries. On this basis the trend of accident costs for the State as a whole is consistent with the severity rate. For the period 1927-1931 the compensation cost trend was upward 4.74 percent, the medical cost trend was downward 7.53 percent, and the industrial insurance (total) cost was upward 2.57 percent. The upward total trend, although relatively slight, is due entirely to the compensation costs per man-day of permanent total disability injuries, the trend increasing 87.5 percent over the 5-year period; there is little or no malingering in this type of injury and costs can be reduced only through accident-prevention work. Serious nonfatal accidents in times past have played a similar role at the New Black Diamond; in many cases they are more costly in dollars than fatalities.

CONCLUSION

To the workmen the frequency and severity of accidents are the all-important factors; to the employer, although these factors are also paramount, safety work is being economically successful only when it reduces the cost per unit of production. The coal-mining industry of Washington for the 5-year period 1927-1931 has reduced its industrial insurance costs (compensation and medical) per unit of production. The New Black Diamond mine for the period 1928-1931 has reduced its accident costs on all bases. Accident-prevention work, not only in the Black Diamond mine, but also in the coal-mining industry of Washington, is therefore successfully accomplishing its purpose.

For compensation and medical benefits alone, coal-mine accidents since 1913 in Washington have cost the employers some $4,086,000 and the employees $591,000; the administration of these benefits has cost the State some $327,000; the total cost therefore has been approximately $5,000,000. During this 20-year period (1913-1933) 475 lives were lost in this relatively small group of coal mines. Truly the prevention of accidents is a mutual problem, and by concerted effort the industry apparently is advancing well on the way toward paying dividends of various kinds through the elimination of economic waste and misery. In this the Black Diamond mine by its excellent safety performance in recent years, but especially in 1931 and 1932, is setting the pace and is showing that even in pitching beds, coal mining can be done with relative safety.

I. C. 6732

July, 1933

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

RECOMMENDATIONS OF THE UNITED STATES BUREAU OF MINES ON CERTAIN QUESTIONS OF SAFETY AS OF FEBRUARY 3, 1933

BY

The Mine Safety Board

I.C.6732,
July, 1933.

INFORMATION CIRCULAR

UNITED STATES BUREAU OF MINES

RECOMMENDATIONS OF THE UNITED STATES BUREAU OF MINES ON CERTAIN QUESTIONS OF SAFETY AS OF FEBRUARY 3, 1933[1]

By the Mine Safety Board[2]

CONTENTS

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6732."

2 G. S. Rice, chief mining engineer, chairman.
O. P. Hood, chief engineer, mechanical division.
R. R. Sayers, chief surgeon.
D. Harrington, chief engineer, safety division.
C. W. Wright, chief engineer, mining division.

NOTE:- The first 10 recommendatory decisions of the 25 reported in this paper were given in Information Circular 6091 (December 1928); these and the eleventh decision were included in Information Circular 6139 (May 1929) and, with the twelfth and thirteenth decisions, were given in Information Circular 6198 (November 1929). In this paper, reporting all the decisions to date, 25 in number, there is no essential difference in the first part of the explanatory text from the text of Information Circular 6198, which covered the first 13 decisions, other than an exception in the field of application of decision 12; but the second part is new, and relates to the last 12 decisions formulated by the Board and approved by the Director of the Bureau of Mines since the issuance of Information Circular 6198.

INTRODUCTION

Mining men and State officials in their efforts to make mining safer are confronted from time to time with various complicated questions. In recent years these questions have tended to increase with the use of mechanical appliances. Twenty-five years ago electricity had been used but little in mines; today there is practically no other means of transmitting power employed in American mines, except that in metal mines compressed air is used, chiefly for drilling and for operating small portable hoists. In coal mines there has been a great increase in mechanization, which has the tendency to concentrate the work of production in small areas, and to increase the local difficulties of ventilation, also it has intensified locally the hazard from coal-dust in bituminous and subbituminous mines.

The answers to the questions which arise are not always obvious, as is shown by the fact that even able, experienced men differ in their opinions of the best way to overcome a particular mining hazard. The difficulty of solving the problems is also demonstrated by the varying provisions in mine-safety rules, both in official regulations and in the rules of those mining companies that are endeavoring to do more than follow the letter of the law. Such differences of opinion as to the best way of meeting a difficult situation are often found within the same mining organization, and, similarly, varying opinions are held by different members of the Bureau of Mines staff.

In order to reduce differences of opinion and to obtain a generally acceptable decision on any question, a mine safety board was established on June 1, 1924, by the Director of the Bureau of Mines -

> ***to consider questions arising within any divisions of the Bureau that require a definition of the Bureau's collective opinion as to safety practices, safety devices, or safety methods for underground operations or open-pit mining. .... <u>The approved decisions shall form the basis of teaching and policy for the Bureau</u>.

The decisions so far made for public issuance relate chiefly to questions on which there have been the greatest differences of opinion and to the best method of meeting certain special or new dangers. Other decisions will be rendered from time to time. Recommendations of Bureau engineers have already been issued on many general mine-safety problems the answers to which are obvious or on which there is practical agreement among mine-safety men. These recommendations are contained in various publications of the Bureau, and those relating to coal-mine safety are discussed in a handbook entitled "Safety in Coal Mining,"[3] and in numerous other mine-safety publications of the Bureau.

The wording of the decisions makes their application largely self-evident, but a certain amount of additional explanation is here given as to the detailed purposes and reasons for which they have been formulated. All the points of practice covered by the decisions were brought out in a study of reports made for and by the Bureau of mine fires, explosions, and general accidents in mines. As previously mentioned, it was found that the different members of the Bureau staff were not always in agreement as to the best method to recommend for technical handling of a specific complicated problem. Such questions as affect fundamental policies of the Bureau on mine-safety technology are referred to the Mine Safety Board, the members of which attempt to reach a common opinion.

Whenever practicable, a tentative decision which has been unanimously reached by the Mine Safety Board is submitted to members of the Bureau staff at the stations and in the field. Their comments are considered and may lead to changes in the draft. The final draft is submitted to the Director of the Bureau; if the recommendation is approved it becomes the decision of the Bureau on the specific question involved.

The decisions formulated up to March 1, 1933, 25 in number, are presented in the order of their approval. The reasons are given for the formulation of each decision and the application of each when not obvious is explained.

Many of the decisions relate specifically to coal mining because of its numerous and special hazards due to inflammable gas and dust. Others apply especially to metal and other mineral mining and a few are particularly related to the driving of long tunnels. But so far as possible, the wording of the decision is general rather than specific and is made applicable to as many kinds of mining as possible.

The order in which the different decisions appear has no significance as to relative importance. The decisions were made as the specific question or questions to which they relate were submitted - generally by field representatives of the Bureau - to the higher officials of the Bureau.

Definition of terms "Permissible" and "Permissibility"

In the Bureau's efforts to increase safety in mines which require various materials, appliances, and methods of using them, there were devised certain tests and specifications which, if met, entitle the manufacturer to state that such essentials and appliances are approved by the U.S. Bureau of Mines and are "permissible" for use in mines under the limitations prescribed in the approval. This, in the case of machinery, apparatus, devices, etc., is indicated by an approval plate attached to the article and, in the case of explosives, may be printed on the wrapper.

Many of the "permissible" articles, such as miners' lamps, machinery, and appliances using electric power, and materials such as "permissible" explosives are of special use in underground workings having atmospheres which may become explosive by reason of the inadvertent presence of inflammable gas or coal-dust.

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3 G. S. Rice Safety in Coal Mining: Bull. 277 Bureau of Mines 1928 141 pp.

Additionally, for example, permissible explosives must not produce, in the Bureau tests, more than a specified amount of toxic gas per unit of explosive; gas-testing lamps and devices must be accurate within specified limits as well as safe in use in explosive atmosphere; and breathing apparatus and gas masks must provide air for the wearer fit to breathe for a specified length of time.

"Permissible" has a recommendatory meaning. The Federal Bureau has no mandatory powers. The authority, if exercised, to compel the use in underground workings of permissible articles in place of unapproved ones, lies in the officials of the respective States.

Decision 1, relating to miner's lamps in coal mines

The United States Bureau of Mines recommends:

1. In all coal mines the portable lamps for illumination be permissible, portable, electric mine lamps; and also

2. In places where fire damp or black damp is liable to be encountered, a permissible, magnetically locked, flame safety lamp for gas detection, or equivalent permissible device, be supplied to at least one experienced employee in each such place; and

3. Any employee before being supplied with a permissible flame safety lamp be examined by a competent official of the mine to assure the man's ability to detect gas; and

4. All coal mines whether classed as nongassy or gassy in any part, be supplied with magnetically locked, permissible, flame safety lamps, properly maintained and in sufficient number for all inspection purposes.

The recommendations were made for the following reasons:

Open lamps or so-called safety lamps of unapproved kind may cause fires or ignite fire damp and coal-dust, and therefore they should not be used in coal mines. Miners' electric lamps have been so improved in illumination, convenience, and safety, that it is no longer to be considered a hardship in mining to use a portable permissible electric lamp; in fact it is quite the reverse.

However, since no approved electric lamp shows the presence of methane or indicates oxygen deficiency, one experienced employee in every place where such conditions are liable to be found should be provided with a permissible flame safety lamp, or equivalent permissible device, to be used for testing.

Although a permissible flame safety lamp is safe if properly assembled and used, there is a possibility of its being a source of ignition of fire damp if the lamp is illegally opened in the mine or is dropped and the glass broken, or if the lamp is struck by a sharp point or is moved so rapidly as to cause flame to extend through the gauzes. Accordingly the use of the permissible flame safety lamp should be limited to gas testing in the hands of competent and experienced employees who can demonstrate with accuracy and safety the percentage of methane in the air in excess of $1\frac{1}{2}$ or 2 percent. The flame safety lamp also gives information as to the sufficiency of oxygen in the surrounding atmosphere.

Other underground employees working or traveling in well-ventilated places should be provided with portable permissible electric lamps giving an excellent steady light; the good illumination which these lamps provide if maintained in proper condition tends to prevent accidents from falls and other hazards, and the equipment itself is free from the danger of igniting gas and coal-dust.

Decision 2, relating to the kind of explosives to use in coal mining

In the interest of safety the United States Bureau of Mines recommends that for blasting in coal mines, permissible explosives, fired electrically, be exclusively used; and that as an aid to blasting, all coal which is feasible to cut, should be cut or sheared.

The foregoing decision has reference to the character of the explosives employed for blasting in coal mines. It does not preclude the use of other methods, devices and appliances for blasting in coal mines (see decision 12, p.19) which have been approved by the Bureau of Mines and placed on its permissible lists. One of the conditions of permissibility of explosives for use in coal mines is the use of electric detonators of standard make for firing them. Others are: Proper placing of the drill hole and the use of incombustible stemming material tightly tamped.

Nonpermissible explosives such as black blasting powder, including pellet powder, dynamite, and special explosives that will not pass the Bureau's tests for permissibility, readily ignite fire damp and coal-dust and have been the cause of hundreds of explosions. Complaint is sometimes made that permissible explosives are not efficient because they cannot be efficiently used in "blasting off the solid"; this practice, however, is a highly dangerous one, and should not be used.

Where coal is properly undercut, sheared, or overcut, blasting can be done efficiently and safely with permissible explosive. The design of coal-cutting machinery has so improved that nearly every condition in mining can be met by selection of a suitable cutter and a suitable mining method. At present at least three fourths of the coal produced in the United States is cut by machines. It is claimed that in some coal beds blasting with permissible explosives following cutting makes a few percent more screenings than if black blasting powder is employed; even if the claim should be correct for special conditions the Bureau believes that no operator is justified in taking the chances of an explosion disaster by using a nonpermissible explosive.

Decision 3, defining what the Mine Safety Board considers to constitute a nongassy, a slightly gassy, and a gassy coal mine

The United States Bureau of Mines believes that all coal mines are potentially gassy; but for purposes of administration in respect to prevention of explosions and fires, the Bureau recommends the following classification:

Class 1 Coal Mine: A practically nongassy mine in which inflammable gas in excess of 0.05 percent cannot be found by systematic search.

Class 2 Coal Mine: A slightly gassy mine in which -

(a) Inflammable gas has been found* but in amount less than 2 percent in still air in any active or unsealed abandoned workings; or

(b) Inflammable gas can be found, but in amount less than 4 percent, in some place from which the ventilating current has been shut off for a period of 1 hour; or

(c) Inflammable gas can be found** but in amount less than ¼ percent, in a split*** of the ventilating current; or

(d) Inflammable gas enters a split*** of ventilating current at a rate**** of not more than 25 cubic feet per minute.

Class 3 Coal Mine: A gassy mine in which inflammable gas is found in amount greater than specified for a class 2 coal mine.

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* By employing an approved flame safety lamp, with flame drawn low, or by employing an approved gas detector, or by sampling and analysis with an approved gas analytical apparatus.

** By sampling and analysis with an approved gas analytical apparatus, or by employing an approved gas detector.

*** If but one continuous ventilating current is employed in a mine this shall be considered a "split" for the purpose of this definition.

**** Determined by sampling, analysis, and ventilating-current measurement.

General Notes Regarding Decision 3

The inflammable gas found in coal mines is, with rare exceptions, methane. In coal-mining fields where natural gas is found below the coal measures (in lower geologic horizons) by deep wells which penetrate through or near to coal mine workings, there have been instances, fortunately of rare occurrence, of leakage from the well into the mine, which have led to explosions in the mine.[4] Natural gas is chiefly methane - almost always more than 85 percent methane - but it usually contains ethane, propane, and traces of butane; therefore, if the latter components are found with methane in mine air, it is an indication of leakage. The lower limit of explosibility of methane-air mixture when there is turbulance is 5 percent and of natural-gas-air mixtures with about 10 percent ethane and associated hydrocarbon gases is 4.6 percent. The limit therefore varies with the character of mixture.

To determine, under the foregoing specifications, the classification of a specific coal mine in respect to gas, it is advisable that systematic testing and sampling of the mine air in each split of the ventilating current be done at least three times in a period of not less than 72 hours. All tests and samples of the mine air, except one, should show an inflammable-

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4 Rice, G. S., Hood, O. P., and others, Oil and Gas Wells through Workable Coal Beds; Papers and Discussions: Bull. 65, Bureau of Mines, 1913, 101 pp.

gas content less than the maximum limit of the class to which the mine has been or will be assigned. In other words, a tolerance of one test or analysis may be permitted to provide for a mistake or a very exceptional occurrence.

When a new mine is being opened in a coal field where existing mines are generally gassy, it is advisable to assume that similar conditions will be found in the new mine; the development and equipment of the mines should be based upon the expectation that it will be assigned to class 3.

There is no more disputed question than that covered by this decision as to what constitutes a gassy (or gaseous[5]) mine. Only a few State regulations give any specific directions for determination of gassiness other than that of whether or not approved flame safety lamps are required.

A coal mine is popularly spoken of as a "closed-light" mine or an "open-light" mine.

Repeated instances of explosion disasters caused by gas ignition in "open-light mines" have shown that the term is misleading and dangerous in its implication. Moreover, such a method of rating a mine as gassy is not satisfactory partly because there is such a great difference in other requirements - in ventilation, in use of electricity, etc. Also it tends to prevent the adoption of the "closed lamp" by many mine operators who do not like to admit that their respective mines are gassy; although this is not a good reason why the operator should not take maximum precautions, it is a practical consideration. The bureau's plan in this decision would get away from the indefiniteness and confusion by specifying in precise terms what it designates as a "nongassy," a "slightly gassy," and a "gassy" mine. The Bureau's specifications are founded on a study of thousands of analyses of samples of mine air submitted to the Bureau of Mines laboratory at Pittsburgh.

Although this decision specifically refers to defining the gassy or nongassy conditions of a coal mine, its provisions should also be applied to those metal mines and tunnels which encounter inflammable gas. Although such occurrences are relatively rare, serious explosions of gas have occurred in water-supply tunnels driven through shales, as at Cleveland, Ohio, and Milwaukee, Wis., and several disastrous explosions have occurred in tunneling in California. Gas explosions have also occurred in sinking shafts for salt in New York and Michigan and for potash in Texas, as well as in operation of other noncoal mines in several States.

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5 Although the word "gaseous" is most generally used in the United States, "gassy" is considered by the Bureau as more accurate in meaning. Literally a "gaseous mine" would be one not excavating solid matter; whereas a "gassy mine" is one in which the presence of gas is incidental to the mining operation.

Decision 4, relating to auxiliary fans or blowers in coal mines and coursing of air

In the interest of safety, the United States Bureau of Mines, recommends that auxiliary fans or blowers should not be used in coal mines as a substitute for methods of regular and continuous coursing of the air to every face of the mine.

The reason for this decision is the increasing use of auxiliary blowers, which in the United States are nearly always electrically driven, as a substitute for proper coursing of the air. Proper coursing should be done by laying out the mine with double or multiple parallel entries, slopes, or rooms, using regularly spaced cut-throughs or crosscuts to provide successive new air connections close to the faces being driven and then successively closing them by stoppings or doors in such manner as to leave only the inner cut-through or crosscut open for air circulation. In gassy mines, line brattices should be used in each entry, slope, or room, from the last open crosscut to the respective face. There is great danger in the use of auxiliary fans that fire damp will be accumulated by recirculation of the air and will be ignited by an electrical spark. Further, the auxiliary fan as commonly used is intermittently operated - that is, operated only when men are in the heading or room - and bodies of fire damp may collect during times when the auxiliary fan is stopped.[6] Use of electrically driven auxiliary blowers has resulted in a considerable number of explosion disasters.

The term "auxiliary fan" does not apply to a "booster fan." An "auxiliary fan" as here used is a temporary installation and takes only part of the air-current circuit[7] as distinct from a "booster fan" which is a more permanent installation through which is passed the whole particular circuit of air to increase its pressure (negative or positive) so as to overcome resistance, with the object of increasing the volume, and (or) forcing the air in adequate amount through distant workings.

The use of a "booster fan" is rarely justified. In most installations investigated there was recirculation of the air, and where this happens if methane is given off in the circuit there is a corresponding increase in gas content in the air circulated. Explosions have resulted therefrom. The best way, which is ordinarily sufficient, is to improve airways and (or) to install a more powerful surface fan to obtain adequate pressure and volume of circuits of air throughout the mine. A booster fan installation should be contemplated for ventilating a distant extension of a mine only where it is not practicable to put down a new shaft and when, after all has been done that is possible to improve the airways, the air pressure required would be so high as to prevent proper operation of the underground ventilating doors.

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6 The only justification for the use of an auxiliary fan in a properly ventilated coal mine is for emergency work, as in driving a crossheading in rock to make a connection with distant workings, and where it is too far to carry the air properly by a line brattice. In such use the auxiliary fan should be operated continuously 24 hours in the day, the fan and pipe frequently inspected, and the air in the face of the heading tested. It is safer that the fan be driven by compressed air, but if it is electrically driven the motor and switches should be of explosion proof type and an armored power conductor employed; especial attention should be given to thorough insulation of the wiring; and the vicinity further should be fireproofed to prevent the possibility of starting a mine fire from electrical breakdown or overheating of bearings.

7 Greenwald, H. P., and Howarth, H. C., Recirculation of Air and Mine Gas Caused by Auxiliary Fans used in Coal Mines: Trans. Am. Inst. Min. and Met. Eng., Tech. Pub. 110, vol. 76, 1928, pp 164-183.

Decision 5, relating to the prevention of coal-dust explosions by rock-dusting

To prevent the propagation of mine explosions, the United States Bureau of Mines, recommends rock-dusting all coal mines, except anthracite mines, in every part, whether in damp or dry condition. It also recommends that rock-dust barriers be used to sectionalize the mine as additional defense; but these should not be regarded as a substitute for generalized rock-dusting.

NOTE: For detailed specifications as to the kind of rock-dust, amount to use, where to be applied and method of sampling and testing, see Bureau of Mines Serial 2606 and American Engineering Standards Committee "Recommended American Practice for Rock-Dusting Coal Mines" quoted in Bureau of Mines Information Circular 6030.

The rock-dusting method[8] of preventing explosions of bituminous-mine dust (anthracite dust was found not to propagate an explosion) has now been accepted by all the chief coal-mining countries throughout the world. It was recommended after exhaustive testing in the Bureau of Mines experimental mine in 1913. The method was officially approved by France about 1917. It was officially required by Great Britain in 1921, and in Germany in 1926.

The term "rock-dust barrier", referred to in the above decision, is used to designate an arrangement in a coal-mine passageway for holding a specific mass of rock-dust (the amount varying with the surrounding conditions at the location), either in a single container or a group of containers at or near the top of the passageway which on discharge will extinguish the flame of a coal-dust explosion that may arrive from either direction along that passageway. Many of the types of barriers that have been installed in mines have not been successful in tests in the Experimental Mine.

A series of tests of various types of barriers[9] was conducted in the Experimental Mine from 1927 to 1931, and as a result 13 barriers of specific design and dimensions are on the approved list for certain locations and conditions.

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8 The suggested use of noncombustible dust to neutralize the coal-dust was proposed by William Garforth in 1891, and testing began under his direction in 1908 at the Altofts Colliery Yorkshire England. In the United States, testing was begun under the Technologic Branch of the U.S. Geological Survey (later the Bureau of Mines) by G. S. Rice and associates in 1909.

9 Rice, G. S., Greenwald, H. P., Howarth, H. C., Tests of Rock-Dust Barriers in the Experimental Mine: Bull. 353, Bureau of Mines, 1932, 81 pp.

Decision 6, relating to sealing all parts of a coal mine which cannot be kept well ventilated and inspected

> In the interest of safety, the United States Bureau of Mines, recommends that in coal mines all entries, rooms, panels, or sections that cannot be kept well ventilated throughout or cannot be inspected regularly and thoroughly, or that are not being used for coursing the air, travel, haulage, or the extraction of coal, be sealed by strong fireproof stoppings.

In the past the question of sealing has been a much disputed one among mining men. Some have considered that it was highly dangerous to inclose areas in a mine which might fill with a body of gas. The Bureau agrees with this point of view, unless the stoppings are strong and fireproof; and by strength is meant sufficient to maintain the stoppings in place if there should be a heavy fall of roof in an empty or partly empty goaf, suddenly compressing the gas or a gas-air mixture.

On the other hand, the engineers of the Bureau find from extensive experience that in large room-and-pillar mines in which the pillars are either not extracted at all or not extracted for a long period, the old areas are not generally well ventilated and often cannot be inspected because of danger of falls, and that the lack of ventilation is inevitable from the enormous territory which may be opened. Furthermore, these regions act as places for accumulating dangerous dust as well as gas.

The Bureau believes that the ideal system of room-and-pillar coal mining is to take out only a small percentage of coal on the advance, generally less than 20 percent, and then to extract the pillars promptly on a systematic line of retreat for the whole mine or for a large panel, which may be considered equivalent to longwall retreating. Under such conditions there should be no need of sealing extensive areas of open unused workings when they can be properly ventilated.

Where the workings of a mine, whether it is a metal or coal mine, have been connected to an adjacent mine and after more or less extraction of the mineral in that part of the mine, the connecting workings are not tightly closed, strong fireproof doors or stoppings should be erected in such connections at or near the boundary of the mine. The objective of this is to prevent fire or fumes entering from one mine to the other. In the case of coal-mine workings which have been connected, these stoppings should not only be fireproof but should be strong enough to resist the pressure of a gas or dust explosion. Explosion tests made by the Bureau of Mines on the strength of stoppings[10] indicate that a plain concrete slab acting as a flat arch when properly keyed into the ribs and floor, probably is the most practicable form of construction for a fireproof stopping strong enough to resist an explosion of ordinary degree of violence.

In erection of barrier-pillar stoppings which subsequently may be subjected to a high hydraulic pressure through abandonment of the adjacent mine workings and their filling with water, a release to avoid high-water pressure should be provided. This might be an inclined pipe extending down into a water-filled sump on the inby side of the stopping, thus furnishing a water seal to prevent escape of gas into workings in active operation.

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10 Rice, G. S., Greenwald, H. P., and Avins, S., Concrete Stoppings in Coal Mines for Resisting Explosions: Bull. 345, Bureau of Mines, 1931, 63 pp.

Decision 7, relating to the carrying of "intake" and "return" air currents in separate shafts, slopes, or drifts

In the interest of safety, the United States Bureau of Mines, recommends:

1. That the main intake and main return air currents in mines be in separate shafts, slopes, or drifts.

2. That the main intake shaft lining be of fireproof construction and there be a minimum amount of inflammable material in or adjacent to the shaft.

The recommendations in this decision are applicable to any kind of underground mining. Bureau of Mines engineers engaged in mine recovery operations during or following fires in coal and metal mines have experienced great difficulty in restoring ventilation and coping with hazardous conditions caused by placing of the "intake" and "return" currents, separated only by a partition, in the same mine opening. Even if the partition is fireproof and remains essentially intact, the smoke and fumes at the bottom of the shaft or slope will almost inevitably work around into both compartments. Also entrance to either compartment at the surface may be made impossible by flame or smoke.

If a strong explosion in a coal mine reaches a shaft in which a partition separates the intake and the return currents, the partition, even though made of concrete, is sometimes blown out because of the unbalanced explosion pressures in the separated compartments of the shaft.

Generally the two compartments in the same mine opening have been used because they were convenient and possibly reduced the initial expense of opening the mine, but such considerations should have no place in the development of a mine, the lifetime of which may be many years.

A fireproof lining in the main intake shaft is recommended to provide a safe means of exit in the event of an explosion or fire. In the greatest mine-fire disaster that has occurred in this country, in which 259 men were killed, the fire igniting the lining of the intake shaft short-circuited to the return shaft and thus prevented the escape of most of the miners.

The best practice today is to have at least three shafts or other openings with fireproof lining and to have the main hoisting shaft nearly neutral, with just enough of a fresh-air intake split to keep it properly ventilated.

It is advisable that there be a distance of at least 200 feet, or as much more as required by State regulations, between the intake shaft and the return shaft and that in this space and (or) immediately adjacent to the shafts, no structures of combustible material should be built or electrical transformer erected or combustible material stored.

Decision 8, relating to definitions used in coal-mine ventilation regulations, but which may also be applied to ventilation in metal and other mines

The United States Bureau of Mines recommends that in mine ventilation the following definitions be used:

1. The term "intake air" and the term "return air" without qualifying adjectives shall be used only to define mechanical movement of the air respectively in an inward or outward direction with reference to the mine as a whole or to any one group of workings.

2. When health and safety are concerned, the term "pure intake air" shall mean -

(a) Air which has not passed through or by any active workings, and (or)

(b) Air which has not passed through or by any inactive workings, unless these are effectively sealed, and

(c) Air which is free from poisonous gas and by analysis contains not less than 20 percent oxygen (dry basis) and not over 0.05 percent of inflammable gas.

In the State mining codes there is much ambiguity as to the meaning of different mining terms. Therefore, as a prerequisite to the formulation of revised or new codes, it is advisable to employ specific terms.

<u>Decision 9, relating to the quantity and quality of air to be furnished in ventilating mines</u>

The United States Bureau of Mines recommends in coal-mine ventilation practice the following specifications as to unit quantity and quality of air:

1. The quantity in cubic feet of pure intake air flowing per minute in any ventilating split shall be at least equal to 100 times the number of men in that split.

2. The quantity of air entering each unsealed place shall be at least 200 cubic feet per minute and as much more as may be necessary properly to dilute and carry away inflammable or harmful gases which may be present.

3. The air shall be made to circulate continuously to the face in every unsealed place into which an appreciable amount of methane enters.

4 The air in any unsealed place shall be considered unfit for men if it shall be found to contain less than 19 percent oxygen (dry basis), more than 1 percent carbon dioxide or a harmful amount of poisonous gas.

5. If the air in any unsealed place, when sampled or tested in any part of that place not nearer than 4 feet from the face and 10 inches from the roof, shall be found to contain -

(a) More than 1½ percent of inflammable gas, the place shall be considered to be in hazardous condition and require improved ventilation, and

(b) If more than 2½ percent of inflammable gas is found, the place shall be considered dangerous, and only men who have been officially designated to improve the ventilation and are properly protected shall remain in or enter said place.

6. If the air in the split which ventilates any group of workings contains more than 1½ percent of inflammable gas, these workings shall be considered to be in a dangerous condition and only men who have been officially designated to improve the ventilation and are properly protected shall remain in or enter said workings.

This decision, while primarily made for coal mining, contains specifications also applicable to other kinds of mining; it recommends provisions for the minimum quantity of pure air to each mine and in each active or unsealed place in the mine, the maximum allowable percentages of poisonous and inflammable gases, and the amounts of these which are so dangerous as to call for the withdrawal of men from the respective parts of the mine.

The first provision calls for the customary State regulation for a minimum of 100 cubic feet of air to be furnished to every man, but as measured in each ventilating split instead of, as in common practice, measuring at the foot of the intake shaft or entrance. It has been found that in some mines there is a leakage or short-circuiting of air at doors and stoppings so that sometimes only a small part of the intake air reaches the working faces.

The second provision calls for a minimum quantity of air current in not only each active working place but also every inactive or unsealed place.

The third provision calls for circulation of air in each place whether active or inactive, to the face or end, to ensure against accumulations of gas in any part of an unsealed place.

The fourth provision specifies the requirements of air fit for breathing.

The fifth provision specifies a hazardous or dangerous percentage of inflammable gas in a body of air in an unsealed place, states where the tests are to be made to determine the limits of the volume of the gassy mixture, and also relates to the withdrawal of men from a place containing a dangerous percentage of inflammable gas.

The sixth provision specifies the percentage of inflammable gas in a ventilating split which constitutes such a hazard that miners should be withdrawn and kept out of that split until the dangerous conditions have been removed by improved ventilation.

The foregoing provisions for continuous circulation of air to the faces presuppose that mining officials obey the mining regulations of the respective State in which the mine under consideration is situated and in addition that they follow the best practices in coal-mine ventilation such as -

(a) Coursing air by means of double or multiple entries or other openings.

(b) Employing crosscuts between the double or multiple entries or other openings at specified intervals.

(c) Erecting stoppings or doors in every crosscut except the one nearest the face in the respective entries or other openings.

(d) Employing the best type of stopping, which, according to the better State regulations, must be built of incombustible material in crosscuts of the main or more permanent openings.

The character of stoppings other than in the main-entry crosscuts is not usually specified in the State regulations, but it is good practice to use incombustible material - preferably massive concrete - for all main stoppings except those for temporary use. In room crosscuts, gob walls or dirt filling is usually sufficient if kept tight, or tight board stoppings may be used; but curtains should not be used except in connection with line brattices extending beyond the last open crosscut, because of safety and efficiency. Check curtains in the mouths of a group of rooms to prevent short-circuiting of the air current is proper practice but the curtains should be kept in good repair.

The character of the doors used for directing the ventilating current and their operation are specified in some of the better State regulations, but the necessity of having doors in duplicate or triplicate between the main intake and return of the whole current or main splits is not specified, although this is in accordance with the best practice. The Pennsylvania mining law requires that where there is a door on "any incline plane or road whereon haulage is done by machinery" an extra door shall be provided for use in case of necessity; also, wherever a principal door is placed, an extra door shall be provided to be used in case of necessity. (Art. IX, sec. 8, Pennsylvania Bituminous Mining Laws).

The character of the overcasts is specified in some of the State regulations, but the advisability of employing "overcasts" or "undercasts" in place of doors wherever practical to do so is not emphasized as it should be, except in the case of Pennsylvania State mining laws applying to certain conditions, as follows: "In every slope with workings on both sides, an overcast and an undercast * * * shall be provided as a passageway for the use of employees to cross from one side to the other" (art. VI, sec. 5); and in gassy mines, "the return air from each split where from 70 to 90 persons are employed shall be conducted by an overcast into the return airway, which shall lead to the main outlet" (art. IX, sec. 1); also, "in every mine all new air bridges, overcasts or undercasts shall be substantially built of masonry, concrete, or other incombustible material, of ample strength, or shall be driven through the solid strata" (art. IX, sec. 7).

In gassy mines line brattices should be used in headings and rooms to carry the ventilating current close up to the face.

Ventilation regulators are a necessity to proper apportionment of the ventilating current in the different splits. Theoretically if the splits are laid out so that the resistance to the movement of the air in each is identical there would be no need of regulators, which cause a loss of power, but while their employment should be minimized by changes from time to time in the splitting of the air current, the irregular advance of workings in a mine of large production capacity necessitates some extent of use.

The number of men on any one split of air is specified in most State regulations; this number varies from 45 to 100 persons permitted to work in one continuous air current or split. With adequate ventilation as specified in this decision, based on the number of men and the inflammable gas content, 75 appears to be a conservative figure to adopt, where not in conflict with State regulations, for the number of men employed on one continuous air current.

Decesion 10, relating to ways of escaping from a mine

The United States Bureau of Mines in the interest of safety in all underground mines, recommends:

1. That every underground mine shall have two or more ways of escape to the surface, so arranged and equipped that men can escape quickly

2. Such ways of escape shall be so separated by at least 50 feet of natural ground throughout their length that damage to one from any source shall not thereby lessen the effectiveness of the other as a means of escape.

3. (a) Where the way of escape is a shaft which is steeper than 45° from the horizontal, it shall have incombustible walls or lining and contain no fire hazard from the surface to the lowest level.

(b) Where the way of escape is a drift or slope which is inclined less than 45° from the horizontal, the incombustible walls, or lining, and freedom from fire hazard shall extend at least 200 feet from the entrance.

4. Where but one way of escape has incombustible walls or lining, it shall be the normal intake airway.

5. Not more than 10 men shall be employed in any part of a mine on any one shift until a second way of escape from that part has been provided.

Decision 10 is intended to apply to any kind of underground mine.

The first provision is one which is to be found in practically all State mining codes. Since the establishment of the Federal Government mine-safety work in 1908 there have been numerous accidents in places from which men were unable to escape because there were not adequate arrangements for their getting out quickly, as by hoist in a deep shaft, or because escape passages were not well timbered, free of debris, and ventilated by intake air currents. For recommedations regarding main intake and return air currents to be in separate shafts, slopes, or drifts and the main intake shaft lining to be of fireproof construction, see decision, page 12.

The second provision is to establish a minimum distance between ways of escape to insure that a fire or explosion which blocks one way would not at the same time make the other means of escape difficult or impossible of access. For recommended minimum distance between main intake and return shafts see page 12.

The third provision relates to making fireproof the shaft, drift, or slope used as an escapeway, the obvious purpose of which is to prevent the respective linings from getting on fire and cutting off the escape of men in the mine.

The fourth provision is to insure that where only one escapeway is fireproofed, that should be used as the intake entrance of the mine. This provision is also covered in a different manner by the second clause in decision 7.

The fifth provision is to prevent the employing of any considerable group of men in any part of any mine where there are not two ways of escape, and limits the group to 10 men on any one shift exposed to a hazard, until a second way shall have been made by sinking shafts or slopes, or driving drifts or crosscuts from the surface or from other workings.

<u>Decision 11, relating to haulage and hoisting in coal mines with reference to the ventilation</u>

In the interest of safety, the United States Bureau of Mines, recommends that in coal mines, haulage and (or) hoisting be kept in intake air as far as possible.

The advisability of this recommendatory decision has been indicated by many reports by Bureau of Mines engineers on mine explosion and fire disasters. Since the general introduction of permissible miners' lamps and permissible explosives in gassy and dusty coal mines, and since, on the other hand, the almost universal adoption of electricity in this country for underground machinery and haulage, the chief cause of explosions in recent years has been electrical. Methane accumulations have been ignited by electric arcs either at or near the working faces, in poorly ventilated places and (or) in return airways, and the resulting explosions, often small at the origin, have been spread disastrously by coal-dust. Although coal-dust can be neutralized by rock-dust as specified in decision 5, a small percentage of methane in the air greatly increases the hazard of ignition and propagation by coal-dust and may require so large an increase in the amount of rock-dust that for high percentages of methane in the air current of a gassy mine, effective rock-dusting may be exceedingly difficult to maintain.

The hazard of ignition of gas or coal-dust is greatly lessened if all machinery and equipment employed in coal mines are of permissible kind.

Trolley haulage, which is so generally employed, and other nonpermissible machines and hoists which cause electric sparks should never be used except in a positive current of pure intake air.

The fighting of mine fires may be especially difficult if the hoisting and haulage is not done in intake air.

Decision 12, relating to methods of firing shots in coal mines

The United States Bureau of Mines extending Mine Safety Decision 2, recommends that for blasting either coal or rock in coal mines, permissible* explosives or equivalent permissible device be used exclusively, and in addition recommends that in blasting** -

1. Each charge shall be in a hole properly drilled and stemmed with incombustible material.

2. Each shot shall be fired separately by a permissible single-shot blasting unit, using an electric detonator or igniting equivalent of a kind specified by the Bureau for the particular permissible explosive or permissible blasting device.

3. Before and following each shot in gassy and slightly gassy coal mines, examination for gas shall be made with a permissible flame safety lamp or permissible equivalent*** and -

4. If more than 1½ percent of inflammable gas is found, in the quantity and by the method specified in Mine Safety Decision 9,**** the place shall be considered to be in a hazardous condition and before another shot is fired the gas shall be reduced by ventilation below the percentage and quantity specified in decision 9.

5. Each shot employing explosives shall be prepared and fired by or under the immediate supervision of a man having a state certificate as a mine examiner, fire boss, or foreman; and whenever conditions permit all other men than those authorized to prepare and fire shots shall be out of the mine when shot firing with explosives is being done.

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* Anything that has successfully passed scheduled tests and is officially approved by the United States Bureau of Mines is termed "permissible."

** Exception: This rule would not apply where shot firing is done electrically from the surface when all the men are out of the mine.

*** Mine Safety Decision 1 relative to permissible flame safety lamps or equivalent.

**** Decision 9, paragraph 5: If the air of any unsealed place when sampled or tested in any part of that place not nearer than 4 feet from the face and 10 inches from the roof shall be found to contain -

(a) More than 1½ percent of inflammable gas, the place shall be considered in a hazardous condition and require improved ventilation and -

(b) If more than 2½ percent of inflammable gas, the place shall be considered dangerous, and only men who have been officially designated to improve the ventilation and are properly protected shall remain in or enter said place.

This decision, as stated in its opening words, supplements Mine Safety Decision 2 (p. 6) which primarily recommended the use of permissible explosives. Since the time of issuing that decision, a blasting device has been tested by the Bureau of Mines and determined to be suitable for use in gassy and dusty mines and has been termed a "permissible blasting device." This instrument consists of a reusable steel cylindrical shell which is charged with highly compressed or liquefied carbon dioxide and contains a heating element to be ignited by an electric squib within the container. A firing circuit can be established only after inserting a bayonet-locked firing plug in the end of the container. Only a permissible single-shot blasting unit should be used for firing.

Permissible explosives are "permissible" only when used in the manner prescribed in the "Schedule of tests." This decision (12) specifies that permissibility requires the

shot hole to be properly located with reference to the "burden" of the blast and that the explosive charge be stemmed with incombustible material. The charge should be fired by electrical detonator, using a permissible single-shot blasting unit, one shot at a time, except in firing electrically from the surface when all men are out of the mine, as is the practice in some of the mines of Utah and in certain other districts.

The rapid firing of a series of shots in coal mines by shot firers who may not have charged the shots, who proceed from place to place firing the groups of shots, and who do not make any inspections presents a great hazard. Where several shots are fired in one place, gas may be liberated and (or) coal-dust stirred into the air by the first shots, and a subsequent shot may ignite the dust and (or) gas, as has frequently happened.

Furthermore, if one or more of the shots in a face or pillar are what are termed "depending shots," it is impossible to foresee how much or how little coal is thrown out by the first blast, leaving too light a burden for the second blast or else so heavy a burden that it may cause a blown-out shot. No permissible explosive or blasting device so far tested and given permissibility is absolutely free under all conditions of use from the possibility of some external flame, and if there is inflammable gas in explosive proportions in an amount in excess of that given in clause 4 of decision 12, or if there is a dense heavy cloud of coal-dust present, ignition may occur which may lead to a disastrous explosion. Although this combination of circumstances is perhaps rare, it has occurred and may occur again if every precaution is not taken.

Special Conditions Which May Require Exceptions to Single Shot-Firing

Where more shots than one are to be fired in a working place of a mine it is possible that under some conditions special hazards may exist in firing shots separately and inspecting between shots; in such cases simultaneous multiple shot-firing may be allowable, when permitted by the respective State mining regulations. This condition may occur in steeply pitching workings or in working places where outbreaks of gas may be released by shot, or where the roof, draw slate, or coal is so friable that despite use of usual good timbering methods, falls are likely to occur after a shot, making it dangerous to inspect the roof and to connect the firing leads for the next shot. This possible exception making choice of a lesser hazard does not mean that the Bureau of Mines recommends simultaneous multiple shot-firing when it is at all practicable to fire safely one shot at a time with careful inspection for gas and roof conditions before and after each shot. When multiple shot-firing is necessitated by the conditions, it should be done by firing simultaneously with standard electric detonators. The use of fuse and (or) delayed-action detonators, so called, is dangerous because of the possibility that a delayed-shot may ignite gas or a coal-dust cloud thrown out by a previous shot.

No multiple shot-firing device for use in gassy coal mines under this exception has been approved by the Bureau of Mines at the time of completing this paper (May 1933), but an investigation is being conducted to determine its feasibility; meantime, when multiple simultaneous shot-firing is necessitated by the conditions, such available multiple shot-firing device should be selected as will give as low-tension current as possible for firing not to exceed six shots at a time, with duration of current of the smallest fraction of a second. Under no circumstances should the power lines be used in blasting when men are in the mine.

In section 1 of the decision (12), emphasis is properly placed on a shot-hole being properly drilled and stemmed with incombustible material." This applies to blasting with a permissible device as well as with explosive.

The only blasting device so far (May 1933) approved by the Bureau contains in addition to liquefied carbon dioxide a "heating element" which might, under certain conditions of imperfect stemming, or if the shot-hole intersects a crevice or joint plane in coal or rock, produce sufficient external heat to ignite coal-dust or inflammable gas. In fact, engineers of the Bureau have observed an "illumination" from shots with this blasting device. Moreover, the sudden release of high-pressure gas, whether it be carbon dioxide or other gas (or air recently tried in blasting devices as yet not tested by the Bureau) might, under exceptional conditions produce sufficient heat by adiabatic compression at some point in the shot-hole to ignite a methane-air mixture. Hence, the need of incombustible stemming for either blasting devices or explosives.

"Properly stemmed" means tightly tamped stemming. The necessity for this for explosives is well understood, but it has not always been understood to apply to blasting devices. Tight stemming is necessary not only to prevent issuance of flame but also to prevent violent ejection of the shells of the blasting device from the hole. A number of accidents have occurred in which men have been killed or severely injured by violent projection of the shell. In certain instances the shell has rebounded from a corner into a crosscut and caused accidents to workers. In one instance in an English mine, the shell was shot through the end of a steel mine car. Even if tight tamping of the stemming is done men should retire to a safe distance and out of the line of fire or rebound, to an adjacent working place. In Great Britain a post, sprag, or other suitable obstruction set up in front of the blast hold, is now required to prevent ejection of the shell in a dangerous manner. Such a method or an efficient shell anchorage within the hold seems advisable in addition to stemming.

Section 5 of decision 12 recommends that the men who fire the shots shall be certified men and that they shall know what each hole contains before firing it, either charging it themselves or having it charged under their immediate supervision. It also recommends as an additional precaution where the organization of the mine and other conditions permit, that shot-firing with explosives shall be done when all other men than those authorized to fire shot are out of the mine.

Decision 13, relating to electrical equipment in coal mines which may become gassy

The United States Bureau of Mines recommends that when electricity is used in coal mines rated as gassy,* or whenever in any mine the atmosphere may become gassy:

1. Electrical equipment shall be permissible,**

2. Nonpermissible electrical equipment*** shall be used only in pure intake air,****

3. Electrical power shall be cut off whenever the air in the workings is in a dangerous condition,***** due to inflammable gas.

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* Decision 3 classifies coal mines on the basis of the specific amounts of methane found in the mine atmosphere. See also, decision 9.

** Anything that has successfully passed scheduled tests and is officially approved by the U.S. Bureau of Mines is termed "permissible."

*** Including trolley wires, trailing cable connections other than through permissible junction boxes, and power lines (except armored rubber-covered cables which meet the specifications of the National Electrical Code.)

**** Decision 8 defines "pure intake air."

***** Decision 9 defines the proportion or amount of gas in a mine working which shall be considered dangerous.

The alarmingly frequent explosions due to electrical ignition by electric arcs from trolley wires and bare power lines, that have occurred in coal mines, especially in recent years, make it highly important that electrical equipment should be "permissible" when used in any parts of a mine where gas is likely to be encountered. The Bureau in Mine Safety Decision 3 states that in its opinion "all coal mines are potentially gassy," but for purposes of administration in respect to the prevention of explosions and fires, it recommends their classification as nongassy, slightly gassy, and gassy.

Decision 9 states that if more than $1\frac{1}{2}$ percent of inflammable gas is found in the air of a place, it is in a hazardous condition, and if more than $2\frac{1}{2}$ percent is found, no men should remain in the place except those charged with the duty of improving the ventilation and who are properly protected, as by wearing permissible oxygen breathing apparatus and using approved gas indicators of dependability and precision.

If the gas inflow into a coal mine from the surrounding strata were constant and the workings and other conditions uniform, provisions for ventilation could be made, except under abnormally gassy conditions, that would practically insure against ever having a hazardous gas condition in that mine, except by breakdown of ventilating equipment. But none of the factors are constant; the gas inflow changes as the mine faces advance, new fissures carrying gas are encountered, large roof-falls occur which tap gas "feeders" or throw down gas collected in caved ground, and the mine workings are constantly changing in shape and conditions, all of which in turn affect the ventilating arrangements.

It is therefore the opinion of the Bureau that the maximum degree of safety from electrical ignition in coal mines would be obtained by using only permissible electrical machinery, and other permissible electrical equipment, including locomotives and underground hoists. The Mine Safety Board recognizes, however, the commercial difficulties of putting this

recommendation into effect, as, for example, in nongassy mines and in the intake air of mines rated as gassy. Nevertheless, the Board is emphatically of the opinion that nonpermissible electrical equipment should not be used except where the atmosphere is as free from inflammable gas as is specified for "pure intake air" in decision 8, which calls for not over 0.05 percent of inflammable gas in "pure intake air." This view is also indicated in decision 11 "that in coal mining, haulage and (or) hoisting be kept in intake air as far as possible."

Power cables for underground use in coal mines classed as gassy have not as yet been covered by a Bureau of Mines schedule of tests for "permissibility." It is tentatively recommended that only those cables termed "armored cables" of the rubber-covered type constructed in accordance with the specifications of the National Electrical Code be used, and that the "armor shall be electrically continuous throughout and grounded." (Tech. Paper 402, p. 5.)[11]

Trailing cables also have not been covered by the Bureau of Mines schedule for permissibility. Until a permissible list for trailing cables is established by the Bureau, the users of permissible machinery should employ the trailing cables recommended for the specific permissible machinery. (Details of the general character, installation, protection, and inspection are given in Technical Paper 402.) In all places where inflammable gas might be encountered and permissible machinery and armored cable is used, the Mine Safety Board recommends that trailing cables should receive current through permissible junction boxes.

In gassy and slightly gassy mines there are great hazards in having trolley wires or unarmored power lines in headings and working faces beyond any open crosscuts. Such electric wires should not extend into rooms, longwall faces, or pillar workings, or beyond any continuously operated ventilating circuit which is not controlled by effective ventilating doors or stoppings. Curtains and loose gob stoppings or any stoppings or doors which allow air leakage should not be considered effective.

One of the greatest electrical hazards, especially in coal mines, is found where pumps with nonpermissible motors and connections and bare cables are used in workings either active or inactive, and especially where the pump has an automatic restarting device controlled from a distant switch. The pumps often are used intermittently and inspected only occasionally. Numerous disastrous explosions and fires have been caused by such pump installations. The hazard exists whether the pumps are in return air or nominally on intake air, as falls of roof may bring down gas or an open door may cause gas to accumulate. Where ordinary power lines are used instead of armored cables, explosions or fires may be started by an electric short circuit in power lines thrown down by a fall, or ignition may occur from sparking of the motor.

11 U.S. Bureau of Mines, Safety Rules for Installing and Using Electrical Equipment in Coal Mines: Tech. Paper 402, 1926, 21 pp. (Sponsored by the U.S. Bureau of Mines and the American Mining Congress.)

Decision 14, clearance spaces for travel in haulageways, shelter holes, and location of trolley and other electrical conductors

In the interest of safety in coal mining, the United States Bureau of Mines recommends:

1. That in all haulageways, there shall be a continuous space for travel on at least one side of the trackway with a clearance of 30 or more inches from the nearest obstruction to the furthest projection of any moving equipment.

2. That any electric conductors other than armored cables, telephone, and low-voltage signal wires shall be placed on the opposite side of the haulageway from the traveling space.

3. That in all haulageways there shall be holes that may be used for shelter, suitably marked, unobstructed, and not over 60 feet apart on the traveling side, not less than 3 feet deep, 5 feet wide, and 6 feet in height or as high as the traveling space if that is less than 6 feet in height.

4. That at sidings of haulageways the tracks shall be so placed that there will be a space between them free from obstruction, at least 3 feet wide between the furthest projections of equipment moving or standing on the two tracks and where necessary because of conditions of roof support, this space may be regarded as a substitute for the traveling space, as provided in the preceding sections.

5. The provision of traveling space shall be as in (1) wherever practicable, otherwise the speed of haulage shall be limited to 3 miles per hour.

Accidents in haulageways due to switching and spragging, coupling, falling from trips, being run over, caught between car and rib, caught between car and roof, and runaway cars cause over one sixth of the total number of accidental deaths in coal mines and a correspondingly large proportion of total injuries. The group of haulage accidents is larger in the aggregate than any group of accidents other than those caused by falls of roof and coal.

In 1931 there were 240 lives lost in haulage accidents in the coal mines of the United States; this figure does not include those electrical fatalities - 33 in number - due to short-circuiting of the trolley wire. The objectives of this decision are to prevent men from being electrically shocked and to provide traveling space and refuge holes in haulageways.

State coal-mining regulations usually require shelter holes on haulageways to be made at intervals of 60 to 100 feet but none of the State regulations specify the minimum width of clearance space between the trackway and the rib or the nearest obstruction. Also, many of the State laws do not specify where trolley or power lines should be placed in the haulageway; the Bureau makes the definite recommendation (section 2) that they should be placed on the opposite side of the haulageway from the traveling space.

Decision 15, wetting machine cuttings, the face, and tops of cars in coal mining

In the interest of safety in coal mining, the United States Bureau of Mines to lessen the coal-dust explosion hazard recommends that:

1. Machine coal cuttings be wet as the cutting is being done.
2. The coal face, and the working place 40 feet therefrom, shall be kept free of coal-dust by the use of water.
3. The top of loaded cars in the working place shall be wet.

Coal-dust is inevitably produced in the cutting, blasting, and loading of coal. The above recommendations are aimed at wetting, so far as practicable, the dust at the point of its formation in the working places and thus prevent it from rising, as a dry dust will do, into the air and being carried by the air current into adjacent places. This decision is not, except for an insignificant overlapping of methods to be used at the faces of the mine, in conflict with decision 5 which recommends rock-dusting of all coal mines in every part; this decision supports that recommendation because if a smaller amount of coal-dust is distributed around the mine, less rock-dust is required for coal-dust neutralization.

Recommendations 2 and 3, advocating wetting the coal face and tops of loaded cars, can best be effected by having water lines extending to each face and watering by hose, each face region being kept supplied with suitable hose.

These recommendations do not apply to anthracite mines the dust of which will not propagate an explosion.

Decision 16, machine cuttings to be removed in coal mining

In the interest of safety in coal mining, the United States Bureau of Mines recommends that:

1. Machine cuttings be removed from the cut.
2. If the machine cuttings are of a character which would contribute to a dust explosion, they shall be sent out of the mine.

The purpose of having cuttings removed from the machine cut is primarily to prevent coal-dust from being blown around the working place in the blasting of coal and, secondarily, to obtain efficient blasting and possibly avoiding blown-out shots by making the cut free from cuttings which support the coal. The removal of the cuttings without unduly stirring dust into the air can be done if decision 15 is followed, by wetting the cuttings before removal.

Recommendation 2 applies to bituminous coal mining.

Decision 17, lessening the formation and distribution of coal-dust in haulageways in coal mines

To lessen the formation and distribution of coal-dust in haulageways, the United States Bureau of Mines, recommends that in bituminous and lignite coal mines:

1. The mine cars should be constructed and maintained dust-tight.
2. The coal should be so loaded that it will not shake off in haulage.
3. The cars and loads should be so sprayed as to prevent dust being distributed along the haulageways.

The recommendations in this decision support those in the preceding Decisions 15 and 16. Although considerable dry coal-dust is always found at the working face even after watering, it is the falling of the coal from cars in haulage and the blowing of dust from the tops of loads by the air currents which are largely responsible for the wide distribution of coal-dust and extensive dust-explosion disasters.

Recommendation 1 calling for tight cars, is aimed at preventing the leakage of coal and coal-dust along the roadway through cracks in the car bodies and around loose-fitting gates. A large proportion of pit cars in the past, as well as at present, permit considerable leakage of coal along haulageways.

Recommendation 2 calls attention to a generally prevalent practice of loading cars high above the top rail of the car so that when the inevitable bumping in haulage occurs, the coal is spilled off, or the topped coal hits cross bars or a low place in the roof and gets knocked off. Much of this loose coal is ground to dust under the wheels of the cars.

The high topping of cars is usually an indication of insufficient car equipment or of inadequate haulage facilities from the working places. Any advantage of increased output from the individual working places by topping is probably temporary and is outweighed by having dirty roadways, a consequent liability to haulage accidents, and an increase of the dust hazard.

Recommendation 3 is aimed at preventing coal-dust on the tops of loaded cars and in empty cars from being blown off by the ventilating current, which is accentuated by the "wind" produced in high-speed haulage.

A recommendation in decision 15 requires that the tops of loaded cars in the working place be wet; recommendation 3 in this decision, No. 17, calls for automatic spraying of the cars at or near sidings, preferably at the entrance to sidings.

Decision 18, superintendents and foremen of coal mines should have State certificates of competency

In the interest of safety in coal mining, the United States Bureau of Mines recommends that:

1. The foreman regularly in charge of underground operations and also any person who, in the absence of the foreman, may be placed in temporary charge should each have a certificate of competency from the State to act as mine foreman.

2. The superintendent or person in responsible charge of the mine, to whom the mine foreman reports, should have a certificate of competency from the State which should be issued upon a showing of underground experience for a period of time as long as that required for a foreman's certificate and upon passing an examination including all technical questions asked in the examination required of foremen.*

3. These certificates should expire after some stated period of time, such as 5 years, and should be renewed only after the applicant has again passed the examination required by the State.

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* Requirements for foremen's certificates will cover this provision.

Experience has shown that while foremen, or mine managers as they are termed in some States, must have State certificates, it is not always required that those who substitute or are in temporary charge have certificates of competency. This is particularly true of the position of mine superintendent, which in many instances is filled by a person who is not a mining man and yet is in real charge of the safety of the employees of the mine and therefore should be competent to direct safely and efficiently when emergencies arise, such as when by reason of fires or explosions the mine foreman may be entombed or unavailable. Further, where there is only a mine foreman or underground manager at a mine and perhaps no other employee with a foreman's certificate, the foreman may spend too large a proportion of his time on the surface when he should be underground directing his working shift. Mining laws in a few of the States which have lesser mining industries do not require certification of fire bosses, foreman, or superintendents. Therefore in these States there is no one officially responsible at a mine for its management, which is manifestly not a desirable condition so far as safeguarding the lives of mine workers is concerned.

Decision 19, concerning internal-combustion engines in underground work

In the interest of safety, the United States Bureau of Mines recommends that:

1. Internal-combustion engines should not be used in any parts of mines and also should not be used in tunnels under construction because of the hazard of carbon monoxide in the exhaust gases, except -

   (a) When the air current is more than 100 linear feet per minute and the toxic gases are always less than 0.02 percent in the air current.

   (b) When the percentage of inflammable gas in the air current is less than 0.25 percent and (or) inflammable gas cannot be detected in any place by a permissible flame safety lamp.

2. Gasoline or other highly inflammable liquids should not be used for engine fuel in mines and in tunnels under construction because of the hazard of their transportation and use.

Many serious accidents have occurred in mines and tunnels through the use of gasoline locomotives, pumps, and underground hoists. In some instances these have been caused by explosion of the gasoline, either in filling the tanks of the engines or in transporting the gasoline underground. A few years ago a disastrous explosion occurred in a mine in the Sarer (France) when a barrel of benzine (gasoline) fell while being lowered in an upcast shaft, the vapor ignited in some unreported way at the surface, and the resulting explosion not only wrecked the surface plant but set fires underground.

There have been many instances of explosions and fires and many cases of suffocation from the exhaust fumes of gasoline locomotives, pumps, and engines in mines in the United States, so that many State laws or mining department regulations have forbidden the use of gasoline engines in mines. However, these State laws or regulations or orders do not in general specifically forbid the use of gasoline motors in tunnel construction.

In driving tunnels for water supply or for general mining purposes in several of the western States, gasoline locomotives (which are in some instances converted automobiles) or gasoline-driven trucks have been used, the exhaust gases of which have caused numerous cases of illness and asphyxiation among tunnel workers.

It has been supposed by many that there was little hazard in the use of gasoline locomotives or trucks in tunnels of large cross-section, but in such tunnels often there is an advance heading in which the locomotive under some circumstances may be used. In other cases the construction of the lining at more or less distance from the face requires extensive forms and staging, usually built of wood, which acts to restrict the ventilation and may bring about a dangerous concentration of inflammable or toxic gases. For example, a gasoline engine might produce locally a physiologically dangerous amount of toxic gases, and (or) if the strata give off methane, as has frequently occurred not only in coal mines but also in tunneling as in places in the Coast Range Mountains of the Pacific coast, an explosive mixture of methane might collect and might be ignited by sparks from the exhaust of the gasoline engine - hence the specifications in the decision requiring a positive adequate ventilating current in which the percentage of toxic gases or of inflammable gas shall never be more than the respective percentages specified.

Although in tunneling it is much safer and generally better to employ permissible storage-battery locomotives and permissible electric pumps and hoists, if the ventilation

requirements are met and other precautions are taken it appears to be reasonably safe to employ Diesel locomotives which use a heavy nonvolatile fuel oil, whereas the use of gasoline or other volatile highly inflammable fluids as a fuel presents too many hazards of fire and explosion to be permitted underground in tunnel construction or in mines.

This decision does not refer to the use of steam-operated locomotives in mines and tunnels. Such equipment was formerly used extensively in mines in thick coal beds in the Pocahontas region of Virginia and West Virginia and also in tunnels that were being driven in different parts of the country, but as many accidents occurred from asphyxiation, steam-driven locomotives have not been used for many years except in a few isolated localities. An accident in which several men were killed by suffocation occurred recently in a mine tunnel in which a steam locomotive was used. Even though supposedly adequate ventilation for the tunneling operation might be provided, the steam locomotive would still present the hazard of fires by sparks or the dropping of hot ashes and hence should not be permitted in mines or tunnels under construction.

Decision 20, relating to ventilation in the development and operation of tunnels, drifts, slopes, and shafts

In the interest of safety in underground mining, the United States Bureau of Mines recommends that:

> While driving tunnels or drifts and sinking or raising shafts or slopes, and also in their operation, there should be an adequate ventilating current wherever men work or travel.

Before starting a mining or tunneling development, plans and arrangements should be made for immediate installation of an adequate mechanical ventilation. Although under favorable conditions tunnels or drifts have been driven and shafts or slopes have been sunk many hundred feet while depending upon natural ventilation, in many instances the absence of mechanically produced ventilation has led to serious accidents. These have been due to inflammable, toxic, or unbreathable gas issuing from the strata, to deficiency of oxygen and most frequently to toxic gases from blasting. Various natural and mining conditions make it unsafe to drive a heading or sink a shaft even a short distance without a positive ventilating current produced by mechanical means.

Therefore, whether hazards are apparent or not at the time of beginning an underground development, adequate ventilating plans should be made and mechanical ventilating apparatus installed before starting. For the simpler shallow shafts or short slopes, steam or compressed-air jets placed in upcast ducts may be sufficient, but in all large undertakings, a reversible centrifugal fan of ample capacity for emergency use and with a good-sized air pipe or duct should be installed. In the more difficult developments, as in long tunnels making much gas, the use of double ducts for intake and return may be advisable, or at the very least the use of a pipe or tube of sufficient cross section to carry an ample air current not only to supply the human requirements of sufficient oxygen but also to sweep away toxic and inflammable gases produced in blasting and coming from the strata.

The dangers of insufficient ventilation in coal-mine developments, from accumulated gases and explosions thereof, are understood by experienced mining men, but the danger of encountering inflammable and toxic gases in tunnels driven through rock or unconsolidated material for water supply, mine drainage, or other purposes is not well known; numerous accidents, however, have demonstrated beyond question that gases in explosive or dangerous toxic proportions are likely to be found.[12] Also, there have been many instances of encountering inflammable and toxic gases in reopening old mines and in sinking shafts through sands, gravels or silt as well as through rock formations of different kinds.

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12 Harrington, D., and Denny, E. H., Gases that Occur in Metal Mines: Bull. 347, Bureau of Mines, 1931, 21 pp.

Decision 21, relating to protection of supporting posts in main haulageways, slopes, and inclined shafts

In the interest of safety in underground mining, the United States Bureau of Mines recommends that:

> In slopes or inclined shafts which are so steep that fallen material may roll or slide, and in main haulageways, posts supporting the roof or sides should be set in protecting niches in the wall. Where this is not practicable, substantial guard rails should be placed along the posts.

As stated in the decision, in slopes or inclined shafts which are so steep that the fallen material may roll or slide, whether these be haulage slopes or inclined shafts, posts supporting the roof or sides should be set in protecting niches or else placed on substantial side walls so that if a fall of roof occurs, the rocks displaced will not, in rolling or sliding, dislodge the legs of timber sets supporting the roof, which might bring down other falls. There have been instances where extensive falls have thus been caused and in certain cases have killed or entrapped men in lower workings.

Derailments of mine cars are likely to occur in any mine. Many derailments which have occurred may be classed as unavoidable because due to a fallen rock or breakage of rail or car wheel or other mechanical part, weaknesses of these types often not being dislosed even by careful inspection. But whether avoidable or unavoidable, serious following accidents may in many instances be prevented if posts, directly supporting the roof and (or) sides and the legs of framed sets, are so placed or recessed in protective niches in the sides of the haulageway that they will not be knocked out by derailment. This may not always be feasible in temporary haulageways but wherever practicable should be done in main haulageways, whether level or inclined; where not practicable, guard rails or continuous heavy planks should be placed along the posts and fastened securely to them in such manner that derailed cars will be fended off and not be likely to pull out the posts or the legs of timber sets.

Decision 22, relating to sealing or guarding openings to mines under certain conditions

In the interest of safety in underground mining the United States Bureau of Mines recommends that:

1. All surface openings to underground workings not necessary for the operation of the mine or for escape, which are not adequately ventilated, timbered, and regularly inspected, should be kept securely closed to prevent unauthorized persons entering and surface fires from extending underground; and

2. All openings necessary for operation or escape should be guarded against the unauthorized entrance of persons.

During the time that mining has been done on a large scale in the United States, hundreds of lives have been lost and many million dollars worth of property has been destroyed through neglect to seal the surface openings of abandoned mine workings. Many of these accidents have been due to men entering unventilated workings to inspect without proper safeguards, and to trespassers entering to dig coal or other mineral and being suffocated by the deficiency of oxygen in the atmosphere of the mine, or poisoned by breathing air containing toxic gases such as hydrogen sulphide or carbon monoxide where there was a smouldering fire in the mine. Many children as well as men have entered abandoned workings or old shafts in a spirit of curiosity, and have been asphyxiated or killed or injured by falls of roof or by falling down a shaft.

The greatest loss of mining property and mineral reserves arising from leaving surface openings unsealed is due to fire. In many instances a small neglected fire has spread through great areas of workings.

One extensive fire in an old anthracite mine working is said to have started from dumping hot ashes into a hole at the outcrop. This fire spread through several hundred acres of old pillar workings, caused the abandonment of many houses on the surface above because of gases and fumes rising through cracks, and was not checked until it reached a solid coal barrier sufficiently deep below the surface to stop it.

Wherever beds of coal outcrop above the water level of the particular locality and there are unsealed openings, fires are liable to occur. The more general causes are spontaneous combustion in slack piles and surface fires, which extend into the mine opening. In some instances these surface fires have been started carelessly by persons igniting brush or building a small fire in the mine opening for heating, and doubtless in other cases the fires have been started maliciously. However, there have been many instances of strokes of lightning igniting brush or timber from which the fire has spread into a mine.

It is evident that the cost of sealing abandoned workings is insignificant as compared with the damage which may occur from fires. Besides the risk of property damage from leaving abandoned or suspended mines unsealed, many instances occur each year of men and children entering old mines and being suffocated.

The question arises as to the responsibility for such occurrences. If a mine of any size is to be abandoned, it is usual to take out the machinery and underground tracks. After that is accomplished the surface openings should be sealed by erecting strong brick or stone walls properly plastered, or else solid concrete stoppings. Where it is thought that the mine may be reopened, steel or concrete doors should be erected and effectively padlocked. In all cases the doors and frames should be of incombustible material.

When it is known that a mine is to be definitely abandoned before the seals are erected it is advisable that the roof for some distance inside be blasted down.

There is another important advantage in sealing drift or slope openings into old mine workings in which the waters are acidic and from which water accumulates and overflows. It has been found by investigations of the Bureau of Mines that acidity of the mine water is largely stopped by excluding the air from the mine and thus preventing oxidation of pyrite, which is practically always found in coal strata. After sealing, the entrance of normal subsurface waters, which are alkaline, will flush out and (or) neutralize the acid salts already formed. The present condition of acid water running out of old coal-mine drifts in some parts of the country has presented serious problems of stream and river polution, notably in central and western Pennsylvania. In some instances the mine owners have been legally compelled to go to heavy expense for construction of diversion dams and tunnels.[13] Hence, the sealing of unused mine workings where it lessens the acidity of waters flowing from the mine has advantages besides those of safety and prevention of possible fire loss.

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13 Leitch, R. D., A General Review of the United States Bureau of Mines Stream Pollution Investigation; Rept. of Investigations 3098, 1931, 7 pp.

<u>Decision 23, relating to checking men into and out of mines</u>

In the interest of safety in underground mining, the United States Bureau of Mines recommends that:

Every underground mine should keep a record in a safe and quickly accessible place on the surface, which will show the time when each person goes into and when he comes out of the mine and which will also show, insofar as practicable, where he may be found while in the mine.

It is the experience of well-organized mining companies that it is an economic as well as a safety measure to have a record made, on a suitable form, of when each man enters the mine. It is advisable that this record include the probable place where the man will be working during his shift; there might be an exception to this in the case of foremen or inspectors or other workers who may be called upon to visit various parts of the mine. Entry should also be made on the record as to the time when each man comes out of the mine.

It is usual to have a record of some kind made by a clerk at or adjacent to the lamp room where safety lamps or electric lamps are issued. At some mines a man receiving a lamp is given a numbered metal check the duplicate to which is hung on the check board at the same time. When the man comes out of the mine he surrenders his lamp and check and the time is noted, or should be, on the record.

Where such records are kept, if a man should not come out of the mine at the end of his shift the unclaimed check would give warning of a possible accident to him and a search for him could be promptly and properly directed. If the mine were so unfortunate as to experience an explosion or fire, an inspection of the record would give knowledge as to those who might be in the mine and where they most likely could be located by the rescue crews. In some cases the checking system has been made ineffective because the check board containing the record was so located that it was destroyed by the force of an explosion in the mine, or because the record (check board) was cut off from access through being located in mine workings filled with smoke and fire gases at time of mine fire. For these as well as other reasons it is desirable that the record be kept in a <u>safe</u> as well as a convenient place.

Decision 24, relating to guarding trolley and bare power lines

In the interest of safety in underground mining, the United States Bureau of Mines recommends that:

1. Trolley and other bare wires should be adequately guarded wherever the wire is less than 6½ feet above the level of the top of the rail;

2. In mines using loading chutes on one side of the haulageway, the trolley and other bare power wires should be placed on the opposite side and adequately guarded at the chute.

3. Trolley and other bare power lines should be sectionalized by electrical switches which can be safety handled, at the entrance to different parts of the mine; and

4. Trolley and other bare power lines should be adequately supported by strong, insulated hangers so spaced that if a hanger is missing the wires will not sag or shift and thus constitute a hazard.

Shocks and fatalities from electric power lines in mines have been one of the major classes of accidents in both metal and coal mines, especially in recent years; most of these accidents are due to men coming in contact with trolley lines and power lines on haulageways. Although it is advisable that separate passageways or manways for the travel of the majority of the employees be provided in which bare power lines are safeguarded so as to reduce the exposure of most of the employees to electric shocks, it is difficult to provide separate manways in all parts of the mine; hence the advisability of following the specifications for installation of trolley and bare power lines in the above decision. Many mining men seem to consider that guarding of bare power wires, including the trolley wire, at the main intersections or similar places where men pass under them is sufficient protection; but there have been hundreds of fatal contacts with unguarded bare power wires at points where haulage men have tried to couple cars or rerail them, or where the motorman or his helper for some reason or other have come in contact with the bare unguarded power wire. Numerous metal mines and several coal mines have installed and maintained guards for trolley wires throughout the trolley haulage roads and have been amply repaid by the resultant freedom from accidents caused by contact with the trolley wires.

Decision 25, barrier pillars between adjacent mines

In the interest of safety in underground mining, the United States Bureau of Mines recommends:

1. That every mine contiguous to one or more mines in the same vein or bed should, so far as practicable, have separate ventilation, haulage, and escapeway systems, and the underground workings of each mine should be separated by a strong barrier pillar to prevent a fire or an explosion penetrating from one mine to another. Provision should be made to relieve excessive pressure of accumulated water on one side of the barrier.

2. In case mines already are connected and thereafter it is practicable to provide satisfactory separate ventilation, haulage, and (or) escapeway arrangements for each mine, a barrier pillar should be established and the openings which connect the mines should be sealed by strong, tight fireproof stoppings or pack walls.

3. The barrier pillar should be maintained intact until the workable mineral has been completely removed from the contiguous workings; then if permitted by the respective State requirements and property restriction, and if it can be done with safety, marketable mineral in the barrier should be extracted.

Many fires in metal mines or the gases and smoke from the fires have passed from one mine to an adjacent mine with disastrous results to life and property. In coal mines similar serious results have occurred from mine fires and explosions, the latter sometimes having even more disastrous results to life, than fires. These instances have been either because of the absence of barrier pillars or because the connections were unguarded by fire- and explosion-proof stoppings and doors.

Under the act of February 25, 1920, operating regulations to govern coal-mining methods and the safety and welfare of miners on leased lands on the public domain were formulated by the Bureau of Mines and issued by the Secretary of the Interior, 1923. Sections 104a and 104b which include essentially the recommendations in decision 25, also call for a lateral strength of stoppings to resist mine explosions having a pressure of at least 50 pounds per square inch on either side of the stoppings. Results of the tests made by the Bureau of Mines in conjunction with the Bureau of Standards and recommendations made for the construction of stoppings were published in Bureau of Mines Bulletin 345, referred to previously (p. 11).

Justifiable exception may be made to recommendations of sealing all connections through a barrier pillar to an adjoining mine when, under conditions which are liable to arise, there is need of maintaining an escapeway through the barrier pillar and adjoining mine, provided that adequate ventilation and timbering is maintained in the escape passages on either side of the barrier and also provided that steel or reinforced-concrete fire doors in sets of two or three are installed in substantial frames of steel and concrete. Additionally, in bituminous-coal mines, approved "rock-dust barriers"[14] for extinguishment of coal-dust explosions should be maintained in the escape passages on either side of the barrier pillar.

As regards removal of barrier pillars where there is not danger to property and lives, a Pennsylvania legislative act (1929) permits removal when approved by a special commission for each specific case.

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14 See footnote 9.

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Clark

I. C. 6733 June, 1933

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

EVOLUTION OF METHANE DETECTING DEVICES FOR COAL MINES

BY

L. C. Ilsley and A. B. Hooker

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

EVOLUTION OF METHANE DETECTING DEVICES FOR COAL MINES[1]

By L. C. Ilsley[2] and A. B. Hooker[3]

INTRODUCTION

The greatest hazard of early mining was from explosions of methane ignited by open lights. With the advent of safety lamps and forced ventilation this hazard was greatly reduced until the application of electricity to modern mining not only multiplied the sources of gas ignition but the accompanying machines and haulage systems increased the amount of explosive dust, through which a methane explosion is more liable to become a major mine disaster, and thus greatly furthered the need of adequate ventilation to keep down the concentrations of methane.

Adequate ventilation is still the best preventive of such explosions. However, even with a well laid out and controlled ventilating system the amount of air passing through entries and rooms is liable to vary within wide limits because of open doors, falls of rock, or other accidental causes. The percentages of methane under normal conditions should therefore be kept so small that they will not be increased to explosive proportions by short interruptions in ventilation.

Mine ventilation and methane detection are inseparable safety precautions. The increasing importance of ventilation necessitates more accurate determinations of the methane concentrations throughout the mines.

This paper discusses in detail some of the steps in the development of methane detection and specially notes the more accurate detectors now available.

One of the earliest means of detecting methane was by a candle. About 1835, John Buddle, Sr., a noted mining man of his period, when questioned by a Parliamentary Committee as to the safety of using candles, replied:

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6733."

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Yes, I consider it a very great protection, the using of candles. It keeps every man upon the alert; we never can by the use of the Davy lamp judge the quantity of air passing so well as we can do with the candle.

Twenty years earlier, in 1813, Buddle, in a communication by letter, gave the following details as to testing for methane with a candle:

The common pit candles vary in size, but those generally used are forty-five to the pound; the wick is of cotton, and the candle made of ox or sheep tallow; but clean ox tallow is the best.

The mode of trying the candle, as it is called, to ascertain the mixture of inflammable gas, is as follows:

In the first place, the candle, called by the colliers the "low", is trimmed, that is, the liquid fat is wiped off, the wick snuffed short, and carefully cleaned of red embers, so that the flame may burn as purely as possible.

The candle being thus prepared, is holden between the fingers and thumb of the one hand, and the palm of the other hand is placed between the eye of the observer and the flame, so that nothing but the spire of the flame can be seen, as it gradually towers above the upper margin of the hand. The observation is generally commenced near the floor of the mine, and the light and hand are gently raised upwards, till the true state of the circulating current be ascertained.

The first indication of the presence of inflammable air is a slight tinge of blue, or bluish grey colour, shooting up from the top of the spire of the candle, and terminating in a fine extended point. This spire increases in size, and receives a deeper tinge of blue as it rises through an increased proportion of inflammable gas, till it reaches the firing point; but the experienced collier knows accurately enough all the gradations of "shew" (as it is called) upon the candle, and is very rarely fired upon, excepting in cases of sudden discharges of inflammable gas.

This shew upon the top of the candle varies very much according to the length of run, or distance, which the current of air has passed through, before it is mixed with the inflammable gas. The shorter the run of the current of air, before it is mixed with the inflammable gas, the less will be the shew upon the candle when at the firing point, and vice versa.

William Clifford, an American mining engineer, left the following record as to the foregoing practice:

> In early days there were employed in mines a class of persons called "Candle Watchers"; and men became very expert in detecting firedamp with an open light, and in judging generally of its constituent character in this rude way -- the writer well remembers his first lesson in this practice from an old miner.

With such crude ideas as to what constituted safety, it was but natural that one disaster followed another with startling losses of human life.

There is no record as to who first discovered that a flame becomes elongated in the presence of methane, nor is there any record as to when this discovery was made; however, it can be safely stated that the candle flame was undoubtedly the earliest of the many schemes that have been devised for methane detecting. About 1816 the Davy flame safety lamp was introduced into coal mines. Although this lamp afforded a very poor means of illumination, its flame could be used in much the same manner as the candle to indicate the amount of methane present in the mine atmosphere and it was safer. Hence this device can well be regarded as the first safety methane detector.

Since the introduction of Davy lamps, there have been a great many attempts to develop detectors that are more satisfactory than flame safety lamps. This has been especially true during the latter half of the period. These efforts, good, bad, and indifferent, are matters of record in the patent files of the various coal-producing countries. The task has proved difficult, so much so that none of the numerous inventions has completely solved the problem of providing a cheap, reliable methane detector suitable for general use; this is attested by the fact that flame safety lamps are still generally used in American mines for detecting methane.

Flame-Type Detectors

Many of the early inventors continued to make use of the principle of the elongation of a flame in the presence of methane in their search for a better methane detector. Some of the more noteworthy designs will be briefly described.

(a) The Gray oil-burning lamp, designed by T. Gray, H. M. assistant inspector of mines for Great Britain in 1868, was a departure from previous types. It had four tubes with openings near the top of the lamp which brought the air to an annular space below the wick. A sample of the air close to the roof could be collected by this arrangement. The ventilation, however, was poor, and the lamp was later modified to what is known as the Ashworth-Hepplewhite-Gray (A-H-G) lamp, which was similar to the Gray lamp, except that it had three inlet tubes instead of four, and two of the inlet tubes had openings near their base which permitted air to enter the lamp and improved the ventilation. While samples of mine air were being taken, these

openings could be closed by sliding shutters attached to the tubes. A later model of the A-H-G lamp had four tubes instead of three.

(b) The Clowes lamp, designed by Prof. Frank Clowes, University College, Nottingham, England, was an A-H-G lamp with an attachment for producing a hydrogen flame. The oil flame was used except when tests for gas were made; then it was extinguished and the hydrogen flame used. The hydrogen flame burned at the top of a copper tube, and its height was regulated by a valve. The hydrogen flame lengthened in the presence of methane, and by means of a scale for reading the elongation it was claimed that amounts of methane as low as one fourth of 1 percent could be detected.

(c) The Stokes lamp, designed by A. H. Stokes, H. M. inspector of mines, Great Britain, also made use of the A-H-G oil-burning lamp, to which was attached an alcohol vessel. The oil flame was used for general illumination and the alcohol flame for detecting gas. The tube-like metal alcohol container could be screwed into a small opening through the oil vessel. When the lamp was not in use for gas detection this opening could be closed with a plug which was fastened to the lamp with a chain. The alcohol flame was said to indicate as low as one half of 1 percent of methane.

(d) The Pieler, an alcohol-burning lamp designed about 1884 by Pieler, an Austrian engineer, was essentially a bonneted Davy with a very long conical gauze. Surrounding the flame was a short cone-shaped metal chimney. In the bonnet was a glass- or mica-covered slit, through which the height of flame could be seen. A graduated scale helped to determine the amount of methane present. It was claimed that as little as one fourth of 1 percent of methane could be detected with this lamp.

(e) The Chesneau lamp, designed by G. Chesneau, a French engineer, used methyl alcohol fuel to which was added some copper nitrate and ethylene chloride. This lamp was reported to be more satisfactory in operation than the Pieler, not only reaching normal condition in a given atmosphere much more rapidly, but withstanding higher air velocities with safety. It was claimed that the Chesneau lamp would indicate as low as one fourth of 1 percent of methane.

(f) The Fleissner singing flame lamp is representative of alarm types of flame-lamp detector. This detector depends for its action upon certain relations between the burner, sound tube, and flame size that produce a warning note or sound. The burner and sound tube are fixed, and the flame size depends upon the amount of methane or other combustible gas present and the adjustment of the wick height.

To obtain even approximate determinations or warnings it is necessary to adjust the flame in air to the prescribed height. If higher initial wick flame is used the detector will operate at lower percentages; it may even operate with no gas present.

American Practice: Flame safety lamps are in general use in American mines for detecting methane in coal-mine atmospheres. Most of the coal-mining States have regulations in their safety codes requiring the use of methane-detecting equipment. The following excerpt is quoted from the Bituminous Mining Laws of Pennsylvania:

> Detection of Gas: In working places where gas is likely to be encountered, a safety lamp, or other suitable apparatus for the detection of fire damp, shall be provided for use with each machine when working, and should any indication of fire damp appear on the flame of the safety lamp, or other apparatus used for the detection of fire damp, the person in charge shall immediately stop the machine, cut off the current at the nearest switch, and report the matter to the mine foreman.

In general, American lamps are naphtha burning and of standard size; they are equipped with magnetic locks, double gauzes, and a bonnet. Steel and brass frame lamps are mostly used because of their inherent strength, but aluminum-frame lamps are much used by mine officials for testing because of their lighter weight. The lamps are furnished either in the flat- or round-wick models and with either steel or brass gauzes.

A small lamp (Baby Wolf) has been redesigned for distinctly gas-detecting service. It is lightweight, but because it gives very little illumination it is used only with other sources of light such as an electric cap lamp.

Nonflame-Type Detectors

With the introduction of portable electric safety lamps in mines there have been developed various kinds of nonflame-type detectors most of which depend upon electric current from small storage batteries for their operation. The successful makes of these detectors may be placed in three groups: namely, (1) the combustion type; (2) the diffusion type; and (3) the catalytic type. The following are representative detectors of these groups:

Group 1 (Combustion Type).- The Burrell detector developed by G. A. Burrell of the Bureau of Mines, is essentially a U-tube, one end of which terminates in a gas reservoir or combustion chamber and the other end in a gage glass having a calibrated scale. To make a test, water is forced into the gas reservoir by blowing into a rubber tube at the upper end of the gage glass. When the water is allowed to return to its original level it draws a measured sample of the atmosphere to be tested into the combustion chamber. The level of water returning to the 0 percent mark on the gage glass indicates that the full volume of the sample has been taken. The methane is burned from the sample by means of an electrically heated filament. The contraction in volume due to the burning is shown by a new position of the water level in the gage glass which indicates the percentage of methane in the original sample.

The Burrell detector gives relatively accurate determinations of methane percentage when used according to the manufacturer's instructions. About 5 minutes are required for each determination; therefore, it is not adaptable to regular fire-boss service, its chief use being by mine officials in checking fire-boss reports or making mine-ventilation studies.

Group 2 (Diffusion Type).- The N and K (Neufeldt and Kuhnke, German) methane detector depends for its action upon the difference in speeds of diffusion of gases through a porous material.

The essential parts of this detector are an earthenware porous cup, an aspirator bulb for getting the test sample into a chamber around the cup, a manometer to show changes of pressure in the cup, and a source of compressed air for flushing the cup and its surrounding chamber after each determination.

The Ringrose (English) detector embodies the same principle as that of the N and K detector, but the application is quite different. The Ringrose detector has an electrically heated filament in the porous cup which burns the methane as it enters, so that instead of an increase there is a decrease in pressure proportional to the percentage of methane present. Burning the methane in the cup eliminates the necessity of flushing between determinations; that is, the detector is continuous in operation.

As approved by the British Mines Department, this detector is combined with an electric hand lamp; the change of pressure in the porous cup operates a diaphragm contactor which lights a red signal bulb. The contactor is set to operate at a definite percentage of methane. The device is therefore primarily an alarm. Its use as a methane detector is very limited.

Group 3 (Catalytic Types).- The fundamental principle involved is the same for all detectors of catalytic group - namely, that metals of the platinum group under certain conditions act catalytically with methane and produce sufficient heating of the metal to change its color or electrical resistance, or both.

If the metal is in a finely divided state, such as spongy platinum, the action is much more pronounced. Platinum wire is active in high percentages of methane when moderately preheated, about 250° C.; it is active in low percentages only at higher temperatures.

Many catalytic-type detectors have been developed, most of which have proved impractical for mine use; however, a few of these detectors will be mentioned to show some of the applications of the catalytic principle, and three detectors of more successful design will be described in greater detail.

The Aitkin detector had two thermometers. The bulb of one of these was enclosed by a mass of spongy platinum and was thus heated above the other in proportion to the percentage of methane present. This detector did not indicate low percentages of methane and was subject to error in any percentage because of changes in the condition of the sponge platinum.

The Liveing detector had two coils or filaments of platinum wire of equal resistance. One of these coils was mounted in a gauze-protected cylinder with a glass end. The other was mounted in an air-tight cylinder with a glass end which faced that of the first cylinder. In pure air both coils glowed with equal brightness when an electric current was passed through them, but when the gauze-protected coil was in an atmosphere containing methane it glowed with increased brightness according to the proportion of methane present. A photometric screen moving over a suitably calibrated scale was used to obtain light balance between the two coils, and the methane content of the atmosphere could then be read directly from the scale. The device was said to be sensitive to methane, but was unsatisfactory as a detector because of the delicate adjustments required. Repeated heating of the exposed coil caused its electrical resistance to change; therefore, periodic zero setting of the scale in fresh air was required.

The Martienssen detector is a visual type in which the percentage of methane is estimated by the color of a heated platinum filament. The filament is a No. 40 gage platinum wire mounted as an inverted U. The top or middle portion has a partial coating of porous metallic palladium, making a roughened coating of 2 to 5 times the diameter of the wire.

When connected to its 2-volt battery and surrounded by air, the legs or uncoated parts of the filament are heated to a dull red, but the coated part, because of its greater conductivity and radiating surface, is not visibly heated. In 1 percent of methane the coated portion is active catalytically and becomes dull red. Its temperature rises with the percentage of methane. Conducted heat also increases the temperature and coloring of the filament legs. Thus there are combinations of color for every percentage of methane, and one who is skilled in the use of this detector can readily detect methane concentrations of 1 to 8 percent and estimate the lower percentages to the nearest one half of 1 percent, which is an accuracy above that of the usual flame-type detector.

This detector proved safe in laboratory tests. It is compact and portable, weighs only 2 pounds, and seems to offer commercial possibilities.

The Ralph detector was one of the first types in which heating and change in resistance of the platinum filament was shown directly by the galvanometer of a Wheatstone bridge. The successful performance of this detector was limited by lack of sensitiveness in small percentages and variations caused by poisoning of the filament by coal dust or sulphur compounds, both of which defects were eliminated in more recent detectors by operating the filament at higher temperatures.

<u>The U.C.C. detector</u> is one of the later types that operate on the Wheatstone bridge principle. The bridge with its galvanometer adjusted to zero in air, becomes unbalanced when in a methane mixture because of an increase in the temperature and resistance of the platinum filament which is one leg of the bridge. The change in the filament is proportional to the amount of methane present and is shown directly in percent methane on the galvanometer.

The detector consists of four parts, all connected by flexible electric conductors. The parts are (1) the detector head containing the heated filament, (2) a meter and bridge compartment, (3) a lead type storage battery, and (4) a cap lamp attachment. All except the detector head are strapped to the user's body. The lighting attachment gives sufficient light for inspection purposes underground.

The detector gives continuous readings; it responds quickly to changes in gas percentage and thus always shows the percentage of gas in the atmosphere that immediately surrounds the detector head. The indications are reasonably accurate in low concentrations, making this detector the first practical device for determining promptly and efficiently the percentage of methane in the various ventilating air currents of mines.

The detector may be operated continuously for about 5 hours on one charge of the battery. If, however, there is opportunity for rechecking the zero scale reading in fresh air, the filament may at times be turned off and a corresponding increase in operating time secured.

<u>The M.S.A. detector</u> is similar to the U.C.C. detector in principle and in general performance but differs from it in the following details:

The heated filament is inclosed in the meter and bridge compartment and the mixture to be tested is circulated past it by means of an aspirator bulb and a sampling hose. Due to the fact that the filament is exposed to the test mixture at the will of the operator and that the filament soon burns all of the combustible gas from its surroundings if the mixture is not being replenished, a zero scale setting can be made on this detector at any time and place. This permits turning off the filament between successive readings and conserves the battery supply.

There are two models of the M.S.A. detector. One is combined with an Edison electric cap lamp and gets its filament current from the lamp battery. The other is supplied by a battery of two no. 6 dry cells and has no lighting attachment.

Both models weigh approximately the same - 18 pounds. The dry-cell model is complete in one box unit; the storage-battery model has its weight more distributed.

The M.S.A. detector shows the methane concentration directly in percent. It is equipped with an expansion or storage chamber on the aspirator side of the filament chamber. There are several restrictions to the flow of gas through the intake line and filament chamber housing. Thus successive operations of the aspirator bulb, by lowering the pressure in the expansion chamber, cause almost constant flow of the gas sample past the filament. The effect is the same as though the filament were actually in the mixture at the point from which the sample is taken. This makes the detector respond quickly to changes in the concentration at the point of sampling and thus permits ready exploring of mine rooms and air courses. The use of a dryer in the sampling line insures adequate accuracy even in the lower concentrations.

Field Tests

Although detectors such as the U.C.C. and M.S.A. do not solve the problem of placing a reliable detector in the hands of every coal miner, they have proved valuable in the hands of skilled fire bosses and mine officials in that they make possible an accurate and quick determination of the condition of a mine atmosphere and thus form the basis for greater efficiency and safety in mine ventilation.

Recently, George S. McCaa, State Mine Inspector for Pennsylvania, used a M.S.A. type of methane detector in making a detailed study of the mine atmosphere in outlet returns in 13 bituminous-coal mines. Nine of these mines were classified as gaseous, and four as nongaseous mines. Data resulting from his study are shown in the following tabulation:[4]

Methane Analysis - Readings at Outlet

Mine No.	Air-sample analysis	Methane detector reading	Cubic feet of methane in 24 hours	Check samples	
				No.	Average error
1	0.13	0.15	500,000	8	0.05 Low
2	.10	.11	225,000	5	.02 "
3	.11	.08	80,000	1	.03 "
4	.09	.08	218,000	2	.02 "
5	.01	.04	1,800	1	.03 High
6	.19	.17	820,000	3	.03 Low
7	.35	.32	605,000	4	.02 "
8	.43	.45	1,050,000	6	.04 "
9	.09	.06	270,000	3	.03 "
10	.18	.20	700,000	11	.03 "
11	.03	.07	5,250	1	.04 High
12	.07	.08	22,000	2	.01 "
13	.01	.01	-	-	-
-	.02	.02	864	2	.00

4 Taken from a paper read by McCaa at the December 1932 meeting of the Coal Mine Institute of America, Pittsburgh, Pa.

The following data from tests of mine returns by Bureau of Mines engineers using U.C.C. detectors show results comparable to those obtained by McCaa with an M.S.A. detector:

Results of U.C.C. detector tests of mine-return atmospheres

Mine No.	Return	Analysis (percent CH_4)	Detector reading (percent CH_4)	Error (percent CH_4)
1	Cross mains	0.40	0.45	+0.05
1	do.	.25	.27	+ .02
1	2 and 3 west	.10	.02	- .08
1	Split 2	.00	.04	+ .04
1	1 northwest	.10	.04	- .06
1	19 butt	.00	.09	+ .09
1	West side	.20	.21	+ .01
2	South side	.38	.42	+ .04
2	North side	.35	.31	- .04
2	Split 4	.55	.63	+ .08
2	Split 3	.50	.55	+ .05
2	Splits 5 and 6	.25	.43	+ .13
3	Split 4	.07	.08	+ .01
3	1 butt	.30	.31	+ .01
3	1 return	.20	.14	- .06
4	6, 7, 8, and 9 returns	.30	.43	+ .13
4	6 west	.50	.62	+ .12
4	Main south	.25	.23	- .02
4	Main north	.20	.13	- .07

CONCLUSION

It is interesting to review the evolution of methane-detecting devices from the safety aspect. The open-flame method first used was extremely hazardous and is unthinkable today as a safe manner of detecting gas. The unbonneted Davy lamp when properly assembled was safe in low-velocity atmospheres, but would be unsafe in velocities used in present-day practice. Later modifications of the Davy lamp were safer and the lamps used at present in American mines are designed to withstand velocities up to 2,500 feet per minute - a value approximately four times that which was safe for the Davy lamp. The Burrell, the U.C.C., and the M.S.A. detectors are all considered safe in high-velocity atmospheres, and in view of their mechanical construction are probably safer than devices having an open flame protected by a glass cylinder bonnet and gauzes, as is the case of the usual flame-lamp type of detector.

In comparing the various detectors they should be grouped as to types of service or application.

Group 1.- Flame-type detectors and detectors such as the Martienssen, that do not permit of accurate determinations but are simply devices for detecting the presence of methane and of estimating its percentage, may be called "detectors."

Group 2.- Those detectors that have calibrated scales through which more accurate determinations can be obtained may be called "indicating detectors."

Group 3.- Detectors such as the singing-flame lamps and the Ringrose detector should be called "alarms."

Detectors (group 1) are the most widely used. They were the first developed and are still considered the basis of comparisons and of State mine regulations relative to methane detection. As a rule these detectors are inexpensive, rugged, and reliable.

Flame-type detectors give warning of oxygen deficiency, and in every mine their use fulfills the law with respect to methane detection.

There are two different services for methane detectors: (1) The checking of all working places just before workers enter the mine and the testing of working places by machinemen and shot firers previous to and during their operations. (2) Determinations of methane content at all faces and sections of the mine in order to lay out and maintain adequate ventilation.

Perhaps the chief difference in these two services is in the relative number and therefore the total cost of the detectors required per mine. Gassy mines require many detectors for use by the fire bosses, machinemen, and shot firers, whereas one instrument of the indicating detector group may be sufficient.

Another distinguishing difference is the kind of service required. A fire boss has a definite section of a mine to cover in approximately 3 hours. Because of the limited time and long walking distance, devices of the detector group that are lightweight and sturdy and permit relatively quick determinations are suitable for the fire boss. They are also suitable for use by the less skilled machinemen and shot firers.

None of the present devices of the indicating detector group is entirely satisfactory for use by fire bosses, machinemen, and shot firers; that is, they lack one or more of the following desirable features: (1) Low cost, initial and upkeep; (2) quick determination; (3) sturdiness; (4) simplicity. On the other hand, at least two detectors - the U.C.C. and M.S.A. - are very satisfactory for checking the gaseous condition of a mine and improving the ventilation.

Indicating detectors of this type permit ready exploring of all parts of a mine; with them reliable data can be obtained at once, which is much more satisfactory than waiting for laboratory analyses of the fewer samples taken with vacuum sample bottles. Desirable changes in ventilation can be more readily anticipated and made; this tends toward greater efficiency in ventilation and greater safety for every one in the mine.

Devices of the third or alarm group are not used in American mines. The writers believe that under American conditions none of these alarms would prove sufficiently reliable, especially at lower percentages of methane, as they are subject to changes in adjustment, fuel, conditions of contact, etc. Their use in European mines may possibly be explained by the fact that in most of the countries, control of mines and mining equipment is much more rigid and centralized than in the United States. Any alarm device that will not give alarm dependably in less than 2 percent methane would probably have little or no application in American mines.

I. C. 6734

June, 1933

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

METAL-MINE VENTILATION

BY

D. Harrington

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

METAL-MINE VENTILATION[1]

By D. Harrington[2]

Ventilation of underground workings consists of establishment of such control of air currents that the underground workers may work in safety, with maximum comfort and efficiency, and without impairment of health; and that the mine openings may be made subject to such control of air flow as to remove from the workings at ordinary times harmful gases and dusts, and at time of emergency, such as fire or explosion, there may be maintained as much or as little air flow as may be desired covering portions of the mine or the mine in its entirety.

Control of air flow is the keystone of any ventilating structure, and this very essential feature is obtainable only by the installation of mechanically operated fans, together with other ventilating devices such as doors, overcasts, regulators, etc. Every mine, large or small, coal or metal, should from the outset be equipped with a fan. While much has been written about natural ventilation and many claims have been made that in specific mines there is sufficient natural air flow, there are few if any mines, coal or metal, where natural ventilation supplies anything like adequately safe or healthful conditions for underground workers even at ordinary times; and at a time of mine fire or explosion the mines depending upon natural ventilation are practically helpless, and certainly are decidedly dangerous to those unfortunates forced to be in them.

While ventilation has practically always been deemed an integral part of coal mining, metal mines have rarely paid much, if any, attention to air circulation until forced to do so by occurrence of some untoward condition or accident. Yet metal mines actually have as great need of efficient circulation of air as have coal mines. The coal mine must remove the dangerous explosive gas methane, also fumes from explosives, and in occasional places other gases such as CO_2 or nitrogen; metal mines have greater necessity to remove fumes from explosives, frequently have occasion to remove CO_2, nitrogen, and other gases from strata, and even the coal miners' explosive gas, methane, is occasionally encountered; in addition, circulating air currents are urgently needed in many metal mines to reduce the excessively high humidity and

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6734."
Presented at the meeting of the Canadian Institute of Mining and Metallurgy, Toronto, Canada, April 4-6, 1933.

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temperature so frequently found in them but very rarely in coal mines. The immense quantities of minutely fine particles of rock-dust floating in the stagnant air of metal mines, which are very largely responsible for miners' consumption and other diseases so prevalent among metal miners in many regions, could be almost wholly removed by adequate ventilation. The generally accepted conclusion that coal miners have a healthful occupation and live to a ripe old age, and that many metal miners contract diseases such as lead poisoning, miners' consumption, etc., and either die early in life or are incapacitated in middle age, is due almost wholly to the superior working conditions in coal mines, chiefly brought about by ventilation.

Our larger metal mines are readily comparable in many respects to an immense office building or a hotel. The various levels correspond to the floors of the building, except that in a mine they are 100 or more feet apart vertically and in a building are 10 to 20 feet apart; the drifts and crosscuts correspond to the halls and corridors; and the raises, stopes and other working faces to the offices, sleeping rooms, etc.; there is the difference, however, that in the mine there are no openings to correspond to windows to the outside, and only too frequently no openings between working places corresponding to interior doors between offices or rooms, thus leaving only one opening to the working place. In general, the present practice even in the better ventilated metal mines is to cause currents of air to flow along the main drifts and crosscuts corresponding to halls and corridors of a building, with only such amounts of air going into the working places as might be expected to seep by diffusion into the rooms of buildings if all outside doors and windows were tightly closed and the one door to the hall left open. The workers are usually engaged in some occupation, such as drilling, timbering, shoveling, etc., which fills the air with fine dust (the most dangerous kind), or possibly they set off some dynamite to bring down rock or ore, liberating not only clouds of fine dust but also of smoke, laden with poisonous gases; and only too frequently the surrounding walls are damp and have a temperature of 80° to 100° F., or even more, and the stagnant air of the place has a temperature of 80° to 100° F. with a relative humidity of 90 to 100 percent.

The metal-mine underground worker frequently works daily under conditions analagous to those on the hottest, stillest, most humid day which can be found in any of our large cities and, in addition, he frequently has to breathe air laded with fine dangerous dust and with various quantities of poisonous gas, such as CO or oxides of nitrogen from explosives. The underground worker in some metal mines must pause several times daily to wring perspiration out of the few pieces of clothing he wears and even to pour perspiration out of his shoes from time to time. The ordinary condition of the underground metal mine working face often is similar to that which would be produced should all the windows of a sleeping room be closed tightly and air derived wholly from one open door to the hall, the resultant headache to sleepers from such situation being augmented for the miner by dust, gases from explosives and decaying timber, and possibly by high temperature and humidity, alleviated to only a slight extent by release of compressed air.

It appears that approximately in order of their importance the main features affecting air in metal mines are: (1) Movement, (2) temperature, (3) relative humidity, (4) gases, and (5) dusts.

The matter of movement, or velocity, of air from the surface, through the working places, then back to the surface is by all means the most important consideration in effecting adequate ventilation of mines and in protecting the health and forwarding and maintaining the working efficiency of the underground workers.

Temperature of underground air is affected by outside air temperature in varying degree, dependent on depth and extent of workings, air velocities, and other considerations; temperature of mine air is very definitely affected by underground rock and water temperatures, by quantity of air flowing, by oxidation (or decay) of timbers and ores, and by mine fires. It is also affected to a greater or less extent by friction due to velocity of flow, by moving of ground, firing of shots, and heat from lights and from breathing of animals. Heated air obtained from other mines and from electric motors and other machinery may seriously affect the local temperature of air of underground workings.

Relative humidity of underground air is affected to some extent by humidity of surface air, but much more vitally by moisture content of walls of underground workings and especially by water dripping through the air. Quantity, temperature, and velocity of air flowing also ultimately affect the humidity of underground air. Handling of air by small fan and pipe units also locally affects mine air humidity and may be utilized to govern humidity of underground air to a certain extent.

Gases to be found in mine air may come from surface air, from breathing of men and other animals, from lights used, from firing of explosives, from compressed air used with machines or as blowers, from operation of various kinds of machinery, and from various rock or other strata encountered, as well as from mine fires that are active or incipient.

Dusts found in metal-mine air are largely derived from dry drilling, from blasting, from shoveling or mucking, from tramming or dumping rock or ore, timbering, etc. Probably well over 50 percent of all metal mines have siliceous material in ore or containing walls, and hence have siliceous dust, which as far as is known is the most dangerous of all dusts, especially when taken into the lungs in large quantities in the extremely finely divided form thrown into the air by dry drilling, blasting, and mucking. Certain siliceous dusts seem to have far less injurious effects than others of essentially the same composition. Dusts of other than siliceous character, while thought not to be so definitely and immediately dangerous, are nevertheless likely to be harmful ultimately, especially if present in the air in very finely divided particles and in large quantities; the dusts of certain soluble lead ores affect workers through skin absorption as well as through breathing.

Below are some general conclusions of the writer on metal-mine ventilation, after many years of observation:

1. In mining, it appears that ventilation, fire protection and prevention, health, safety, and efficiency are very closely interlocked.

2. There is at least equal reason for providing adequate ventilation for most metal mines as for providing ventilation for coal mines.

3. Metal mines rarely, if ever, make provision for ventilation until forced to do so by some untoward condition or occurrence; coal mines, on the other hand, almost universally provide for ventilation.

4. Efficient ventilation of metal mines consists in supplying at all times such volume of circulating air at places where men work as will enable the worker to exert himself in comfort at maximum physical capacity without endangering his health.

5. Many, probably most, metal mining operating officials are ignorant of the principles of air circulation; this is true of those technically educated as well as of those without technical training.

6. Workers in metal mines, including shift and other bosses, should be educated to respect ventilating devices, such as doors, regulators, overcasts, brattices, fans, etc., as coal miners do, and to become as familiar with those devices as coal miners are. Many present-day reactionary metal miners and bosses consider ventilation a useless fad and obstruct rather than aid ventilation improvements.

7. Ventilation should be under definite, constant supervision, and preferably the person in charge should report to the highest officials, as many local officials in metal mines are not in sympathy with ventilation betterments.

8. Each mine should be ventilated wholly within itself; interventilation of mines is likely to be dangerous, inefficient, and unsatisfactory.

9. Every mine, coal or metal, should have a mechanically driven fan or fans placed preferably on the surface, fireproof housed, and capable of reversing air currents with minimum delay. Metal mines should provide fan ventilation from start of opening in order to avoid dangers from explosive fumes, dusts, heat, etc., and provide fresh air to workers.

10. At seasons of the year when the temperature of surface air and of underground rock and water are about equal, mines relying on natural ventilation are likely to have periods when air circulation is sluggish, or ceases utterly or reverses in direction.

11. At time of mine fire, naturally ventilated mines are likely to be at a decided disadvantage through inability to control the direction of air currents.

12. There are records of naturally or otherwise inadequately ventilated mines filling with CO_2 or other gas, overcoming some of the workers (in some cases with fatal consequences) and compelling suspension of work for considerable periods of time. Upon establishment of efficient mechanical ventilation this situation is readily controlled.

13. Workers in many metal mines are much less healthy than workers in coal mines, due largely to the superior ventilation of the collieries.

14. Miners' consumption, the scourge of metal miners, is caused primarily by breathing very fine particles of certain mineral dusts and especially of siliceous dust. Over 50 percent of our mineral-producing mines are working in more or less siliceous material and through lack of ventilation are giving this dangerous dust maximum opportunity to exercise its harmfulness. Lead poisoning afflicts workers by skin absorption of soluble dust of lead ores as well as by breathing of such dusts.

15. The best remedy for the dust menace in mines, other than preventing its formation, is the universal coursing of currents of air to remove the dust, as it has been proved that the very fine, most dangerous dust in metal mines remains suspended in still air several hours.

16. Metal-mine dust, acting through miners' consumption, lead poisoning, bronchitis, etc., is the chief instrumentality in causing perhaps more deaths annually among the approximately 200,000 metal miners in North America than coal-dust causes to the approximately 700,000 coal miners through explosions.

17. It is fairly well established that miners' consumption is caused chiefly by siliceous dust, but it is probable that any kind of dust in large quantities in finely divided form in mine air will prove harmful to workers ultimately. Investigations in metal mines of the United States indicate that the air of mines so far studied is from 7 to over 40 times as dusty as in South African gold mines.

18. Intake air of metal mines is frequently dusty by having a crusher house or other dust producer near the collar of the intake air shaft, or by having ore skips or cars or ore dumping places in intake air passages without the slightest attempt to allay the dust produced or to prevent its entering the mine.

19. The dustiest, most unhealthful occupation underground is dry drilling, and the average dust content of air in places where this work is done in five large mines in various parts of the United States was 205 mg. per cubic meter of air, yet for similar work in South African mines, but using available

precautions against dust formation, the dust content of air is said to be less than 5 mg. per cubic meter of air.

20. While there are regulations in many of the States of the United States to prevent dust formation in drilling, these regulations are not always lived up to. Miners, while recognizing the dangers from dust, often prefer to take the risk rather than endure the slight discomfort or extra trouble of using precautionary methods or devices; and mine and State officials appear to feel that unless the miner will willingly aid in protecting himself, they cannot force him to protect his own health and incidentally that of his family. Dust-prevention devices of proved success, such as present-day self-rotating wet stopers, should entirely supplant dry drills, and their use should be enforced upon both miners and operators in metal mines, as there is absolutely no valid excuse for dry drilling in present-day metal mining except in a very few instances. To date no workable device is available for removing dust in dry drilling in underground mines.

21. Spraying devices available to reduce dust while drilling may be effective if used intelligently; on the other hand, they may even intensify the air dustiness if used without intelligence, and unfortunately the latter is generally the case. Efficient water drills are now available for all purposes in metal mining (including efficient wet stopers for upper holes), and dry drilling should be prohibited.

22. Some metal mines with high dust production under certain conditions at working faces, have comparatively low dust content of the air in these places at other times, and have low average dust content of air in all places due to efficiency of ventilating currents, especially at working places. Significantly these mines appear to be singularly free of miners' consumption or other diseases, yet the employees work at top speed and the material handled is highly siliceous.

23. The use of poorly placed compressed-air hose blowers at working faces frequently intensifies air dustiness by allowing high-velocity compressed-air streams to pass through dry, loose, finely divided ore or other material being drilled, shoveled, or otherwise handled, and thus forcing workers to breathe this air highly impregnated with dangerous fine dust.

24. While finely divided dust in mines is probably the chief cause of miners' consumption, it is now recognized that there may be other factors of almost equal influence, such as high temperatures and humidities, harmful gases, and lack of air movement; all of these defects are readily remedied by ventilation.

25. It appears that with dry bulb temperature below 75° F., mine working places may be comparatively comfortable, irrespective of air movement or relative humidity. However, the presence of air heavily depleted of oxygen (say below 18 percent) or impregnated with gases such as CO_2, CO, oxides of nitrogen etc., any or all of which may be produced in blasting, may produce uncomfortable or unsafe conditions; also, such places may be both uncomfortable and unhealthful if large quantities of finely divided dust are present.

26. With dry bulb air temperatures above 75° F., comfort and maximum working efficiency can be attained only when the air is moving, this being especially true if the air has high relative humidity. The exact velocity necessary is a variable dependent largely upon the temperature and humidity.

27. Saturated atmospheres, up to nearly blood temperature, may be made endurable, and even to a considerable extent comfortable, by providing sufficient velocity.

28. In still air in metal mines with a temperature of about 85° F. and 90 to 100 percent relative humidity, there is likely to be little effect on persons completely at rest, but upon doing even moderate work body temperature is likely to rise to over 100° F., blood pressure to fall perceptibly, and pulse beat to rise materially. In still air with temperatures of 90° to 100° F. and above 90 percent relative humidity, even when the body is practically at rest, body temperature rises quickly, reaching over 102°; blood pressure is likely to fall rapidly, pulse beat to increase abnormally and to be very sensitive to even slight exercise, perspiration to be very profuse, and dizziness, physical weakness, mental sluggishness, and headache are experienced; upon attempting even light work these symptoms are likely to be greatly augmented.

29. Relative humidity, even up to the saturation point, does not appear to be harmful to health, comfort, or efficiency until temperatures run above 75° F.; and if sufficient movement is supplied, high relative humidity is not particularly harmful until temperatures are well over 90° F.

30. With exception of blind-end working faces, metal-mine air in general is not particularly deficient in quality. However, blind-end faces of drifts, crosscuts, raises, winzes, and stopes in metal mines are likely to have air deficient in oxygen and high in nitrogen or CO_2, and possibly in CO, oxides of nitrogen, or other impurities. There are many cases on record of asphyxiation in metal mines from these gases.

31. Practically all of the explosives used in metal mines give off small percentages of poisonous fumes -- CO, H_2S, or oxides of nitrogen -- when fired, and SO_2 or SO_3 may be found when blasting ores high in sulphur content. If these fumes are not removed from confined working places by ventilation they cause headache, nausea, and even death. The gelatin dynamites give off a less quantity of dangerous gases than do the ammonium dynamites and the latter give off much less than the straight nitroglycerin dynamites; hence, the straight nitroglycerin dynamites should not be used underground, and all places using any kind of explosive should be thoroughly ventilated after blasting and before workers arrive. No blasting of any kind should be allowed during the working shift. Good ventilating currents should also be provided while shoveling blasted ore at the face to prevent workers from being affected by headache, nausea, or other illness from breathing explosive fumes stirred out of muck piles. Compressed-air hose blowers are ordinarily not particularly effective in removing explosives fumes.

32. In stagnant air comparatively small quantities of impurities, such as 0.30 percent or over of CO_2, 0.02 percent or over of CO, or oxygen below 20 percent, cause headache, dullness, etc., and this is particularly true when temperatures are above 80° F. However, these small quantities of impurities are not likely to be very noticeable when there is perceptible movement of the air.

33. Frequently blind-end working faces in metal mines have air so depleted of oxygen that a candle will not burn and carbide lamps must be used; hence, oxygen is below 18 percent, and CO_2 may run several percent; occasionally entire mines are found with this condition, which many metal-mine managers hold to be perfectly all right. There is absolutely no question that men working in an atmosphere which will not support combustion of a candle cannot deliver maximum efficiency and that their health must ultimately suffer.

34. Mines with cool working places which allow men to work at top speed, especially when contracting, are likely to be extremely dangerous to health of workers unless provision is made to remove explosive fumes and other gases and fine dusts from working places by ventilating currents.

35. Mines with high temperatures, above 75° F., and high humidity, above 85 percent, are likely to lose from 25 percent to as much as 75 percent of the efficiency of workers, and workers are likely to become unhealthy ultimately unless moving currents of air are supplied to working places. Unhealthfulness and inefficient results are hastened and intensified if fine dust is present, especially siliceous dust, and if blasting is done, especially when men are in the mine.

36. Many accidents in metal mines are due to deficient ventilation. Failure to remove smoke and fumes from explosives prevents proper inspection of working places to make them safe, and, in addition, many men have been asphyxiated in explosives fumes; in hot, humid, stagnant air men are likely to be affected by dizziness or by lack of ability to think clearly or quickly, or they may faint at an inopportune time and be killed; also, there are numerous instances where men have been known to drop dead due to heart failure in these hot places.

37. When air temperature is over 75° F., the giving of air movement or velocity at working places is of more importance than any other consideration in giving adequate metal-mine ventilation, provided the air is reasonably free of noxious gases.

38. While variations in surface-air temperature and humidity may have a noticeable effect on mines with shallow workings, in general, they change conditions very little if at all at faces in mines with extensive workings; hence, underground working temperatures in large mines vary but little, due to outside air conditions.

39. Air flowing in underground passages rapidly takes the temperature of the surrounding rock. The rate of change is variable and frequently is as high as, or higher than, 1° F. for every 100 feet of travel, even when air velocity is several hundred feet per minute. Still air underground rarely varies more than a few degrees from the temperature of the surrounding rock or water.

40. Rock temperature generally increases with depth, the rate of increase varying from 1° F. or more per 100 feet of depth in certain districts of the western part of the United States, to but ½° or 1/3° F. per 100 feet of depth in other regions, both of the United States and of foreign countries. In Montana, rock temperature in copper sulphide veins is about 108° F., at a point 3,800 feet below the surface, rate of increase being about 1° per 100 feet of depth. In a lead sulphide vein in the Coeur d'Alenes in Idaho, rock temperature 2,000 feet below the surface was but 50° F. In a Michigan copper mine with native copper ore, rock temperature was about 82° F. at a point 5,000 feet below the surface. In a gold-bearing quartz vein in Arizona the rock temperature was 90° F. at a point 600 feet below the surface, while in a quartz deposit bearing copper sulphide in another Arizona district, rock temperature 600 feet below the surface was but 70°. Temperature of coal in place in coal mines in the United States is rarely above 70° F. and is generally much lower. Magazine articles give the rock temperature of the Kolar gold field in India at 118° F. at a point 6,100 feet below surface, and give 98° F. at the 4,000-foot level of the St. John Del Rey mine in Brazil, while it is calculated that at the 8,000-foot level of the City Deep mine in South Africa the rock temperature will be but about 97° F.

41. Rock temperature may vary at the same depth in different kinds of material. A copper sulphide ore with quartz gangue in a mine in the western part of the United States had rock temperature several degrees higher than a zinc sulphide ore in quartz gangue in a parallel vein about 200 ft. distant, both on the same level and both practically free of water.

42. Water standing still or flowing on the floor in mines readily communicates its temperature to surrounding air; water dripping through the air quickly brings the air practically to temperature of the water drippers, and profuse water drippers will determine the temperature of the air almost irrespective of rock temperature. Water temperature underground is generally the same as that of surrounding rock, but not always.

43. Mines having rock temperatures, hence generally air temperatures, above 80° F., frequently have available water piped from the surface which is found underground with temperature below 70° F. Few if any mines take advantage of this water to cool the air by installation of sprays, yet this is a very effective method of reducing the temperature of the air. Mine managers state that they fear that water sprays will cause excessive humidity, forgetting that mines generally have the humidity anyway, and that if the air can be cooled to 75° or below and given a slight movement the high humidity is not harmful. Physiological experimental work in South African mines shows

that men working in stagnant air with a relative humidity of 95 percent and a temperature of 87° F. increased the amount of work performed 46 percent by the mere expedient of installing a small fan to move or stir the air, showing that high humidity in itself is not particularly detrimental, at least until temperatures are well above 90° F.

44. Air passing through fans frequently has the dry bulb temperature increased several degrees Fahrenheit and the relative humidity automatically decreased. In one instance, an underground fan with air delivery of over 20,000 cubic feet per minute, had 8° higher temperature of air at delivery than at intake, the points of measurement being less than 50 feet apart. Similarly, small fan-canvas pipe units used underground for local ventilation frequently have delivered air several degrees higher in temperature than that of intake air at the fan.

45. Small electrically driven fans with galvanized iron or canvas or other flexible tubing are being widely used in metal mines to carry air to dead ends. The galvanized iron has the advantage of allowing reversal of air currents to pull smoke out after blasting, then to force moving air to workers after removal of smoke; moreover, it does not decay as fast as canvas. The flexible tubing must be, in general, used only in forcing air to the face, its advantages being low first cost, ready installation and removal, flexibility in conforming to bends or turns; and ease of repair. Moreover, because of its ready installation and removal, the flexible tubing can be brought close to the working face at ordinary times and easily removed prior to blasting to prevent its destruction. Either method readily admits of placing from 500 to 5,000 cubic feet of moving air per minute at the working face at comparatively small cost.

46. Compressed air from the end of air hose is used to a very great extent to remove explosives fumes from faces or to ventilate hot, stagnant blind-end workings in metal mines. These blowers deliver about 100 cubic feet of air per minute, but its temperature rarely varies much over 2° or 3° from the temperature of the rock and air of the working place. Such compressed-air blowers are inefficient as to removal of smoke or gases, provide comparatively little pure air, and give very little reduction in the temperature of the surrounding air. Moreover, it costs about 100 times as much to place 1,000 cubic feet of compressed air at a working place as to circulate a like amount of air by ordinary ventilation methods.

47. The use of electrical machinery underground causes considerable local increase of air temperature. A magazine article a few years ago described a proposed fan installation at a South African mine to force 75,000 cubic feet of air per minute from the surface into a mine for the sole purpose of ventilating the region around an electrically driven underground hoist, this air to be removed from the mine after passing through the hoist room.

48. Cooling of air in mines is effected by use of ice or sprays of cool water, by refrigeration, by rapid coursing of air brought from the surface or carried through workings with cool walls, and by excluding air from abandoned workings and return air from currents of active workings. Water sprays are not employed nearly as much as they should be; ice is used to a slight extent in the United States but is probably employed much more extensively in South Africa; refrigeration is costly and found only in a very few mines; and the rapid coursing of air currents from the surface, which can be brought about most efficiently by the establishment of definite separate splitting systems, is used only occasionally though it is quick, cheap, and efficient. Failure to seal abandoned places having decayed timber and hot rock or water sends much unnecessary heat into metal-mine air, and reusing of return air has the same effect.

49. At time of fire in a metal mine, lack of an efficient ventilation system may be disastrous. Each mine should have fans which should be so placed as to be inaccessible to fire, have fireproof housing, be capable of quickly reversing air currents if desired, and preferably installed on the surface. There should be a definite system of air splits such that fire in one place may not necessarily fill the entire mine with poisonous fumes. This provision is of vital importance, yet the writer knows of very few metal mines which have made even a reasonable attempt to establish this excellent safety feature. There should also be a system of doors near shafts in the levels leading from shafts such that the entire shaft may be readily isolated in case of fire, or any part of the mine may be isolated from the shaft.

50. Experimental work in mines reveals that after smooth-lining of shafts previously having ordinary exposed timbers there will generally be a reduction of friction to such an extent that 50 or more percent additional air can be handled by the same power. If the smooth lining is done with gunite, it also serves as a fire retardant. Preferably, at least the intake air shaft should be fireproof; and, if possible, all shafts or heavily inclined openings which carry air or through which men travel or are transported should be fireproofed.

51. While the cost of establishing a ventilation system for a large mine is variable, the cost of operation is not particularly burdensome. The operating cost will almost invariably be offset by savings, in compressed air and in increased efficiency and health of employees, which frequently will cover the entire cost of the investment within a few years. If a fire occurs, an efficiently installed and operated mechanical ventilation system is of incalculable value, and the absence of such a safeguard is likely to result in a heavy loss of property and possibly of life.

Return to D Harrington

I. C. 6735

June, 1933

Department of Commerce

UNITED STATES BUREAU OF MINES

Scott Turner, Director

INFORMATION CIRCULAR

COST OF EQUIPPING AND DEVELOPING A SMALL GOLD MINE IN THE BRADSHAW MOUNTAINS QUADRANGLE, YAVAPAI COUNTY, ARIZ.

BY

David C. Minton, Jr.

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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

COST OF EQUIPPING AND DEVELOPING A SMALL GOLD MINE IN THE BRADSHAW MOUNTAINS QUADRANGLE, YAVAPAI COUNTY, ARIZ.[1]

By David C. Minton, Jr.[2]

INTRODUCTION

This paper describes in detail the cost of equipping and developing the Golden Belt mine, Yavapai County, Ariz. These costs should be typical of the average small gold mine in central Arizona which has been equipped with used machinery at a minimum of capital expenditure. The conditions of remoteness, transportation, water, and power supply are about average for properties in the Bradshaw Mountains and that vicinity. There are several mines in the district in the process of development and several dormant properties that might be opened in the future.

The Golden Belt group of claims was taken over by the present management in May, 1931. Although some active development has been carried on in recent years, resumption of work at the old property required entire rehabilitation of the equipment and camp. As the water level had not been reached, there was no problem of dewatering. In the presentation of costs in this paper an evaluation of the cost of equipment and development work as taken over by the Golden Belt Mines, Inc., has been made. Exact data, however, on previous exploration and construction costs are lacking. The total cost of equipping and developing the mine to its present stage has been obtained by adding the disbursements of the present management to the above-mentioned evaluation. These costs do not include the cost of replacing camp equipment destroyed by fires, of unsuccessful milling experiments, management, or obsolete equipment on the property.

The author acknowledges the courtesy of A. L. Lampton, president of Golden Belt Mines, Inc., in permitting the data in this paper to be published.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6735."

2 One of the consulting engineers, U. S. Bureau of Mines.

LOCATION

The Golden Belt mine is in the Black Canyon mining district, Yavapai County, Ariz., on the eastern slope of the Bradshaw Mountains at an elevation of approximately 3,100 feet. The camp is situated on Turkey Creek, 1½ miles west of the Black Canyon Highway, 2 miles east of Cleator (formerly Turkey Station), and 68 miles north of Phoenix. Mayer, a town on the highway and the A. T. & S. F. Railway 15 miles north of the camp, is the shipping point. Although climatic conditions are ideal for operation throughout the year, heavy rains and snows sometimes hamper transportation by truck. Concentrates are shipped to Mayer by truck in 3 to 4 ton loads, each truck making three round trips per shift. Supplies and equipment are trucked in from either Phoenix or Prescott. Hauling of both concentrates and supplies is done by company truck at a cost of approximately $0.05 per ton-mile.

WATER, POWER, AND FUEL

The water supply for the camp and mill is pumped from a well on the edge of Turkey Creek about 100 yards from the mill. This well furnishes an abundance of water all the year around at a depth of 14 to 30 feet. A short drift from the well shaft beneath Turkey Creek intercepts the creek underflow. The water pumped from the mine is not impounded, although plans have been made for its use for milling purposes.

Power is purchased from the Arizona Power Co., which serves a large portion of northern Arizona. The power rate is 1½ cents per kilowatt-hour, with a standard demand charge of $5 per horsepower for the first 25 hp. and $3 per horsepower thereafter. A refund of 15 percent of the monthly power bill is given on a deposit, which was made to the power company, for the construction of a 6-mile power line. The power is transmitted at 17,000 volts; it is stepped down at the property to 440-volt, three-phase, 60-cycle current for operating mine and mill motors and to 110 volts for mine lighting and domestic purposes. The cost of lighting the mine by electricity is one half that of lighting by carbide lamps.

Fuel for heating and domestic purposes is obtained on the premises. There is no timber suitable for mining purposes, however; mine timbers must be shipped in, but as the ground stands well, the minimum of timbering is required.

GENERAL GEOLOGICAL FEATURES

The predominant rocks of the district are Yavapai or pre-Cambrian schist and Bradshaw granite cut by dikes of porphyry, diabase, diorite, and quartz. The general formation at the Golden Belt mine is schist; the dip of the schistosity is vertical and the strike north and south. The schist is cut by dikes of rhyolite porphyry and altered diabase.

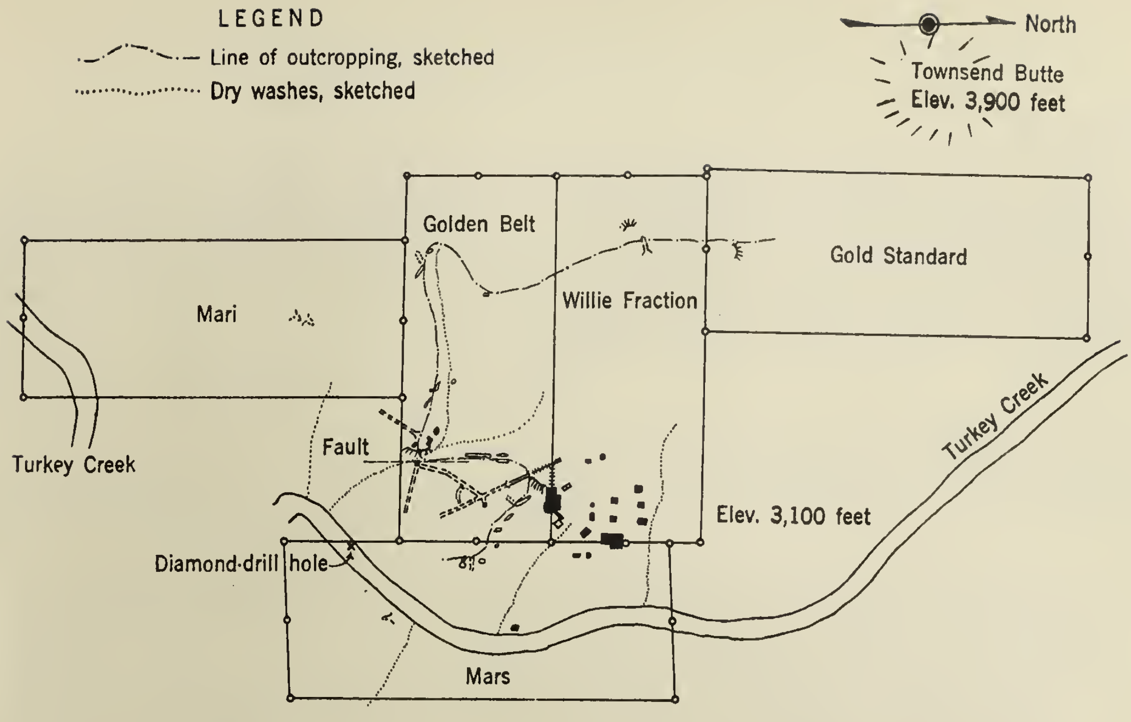

Figure 1.—Golden Belt group of claims.

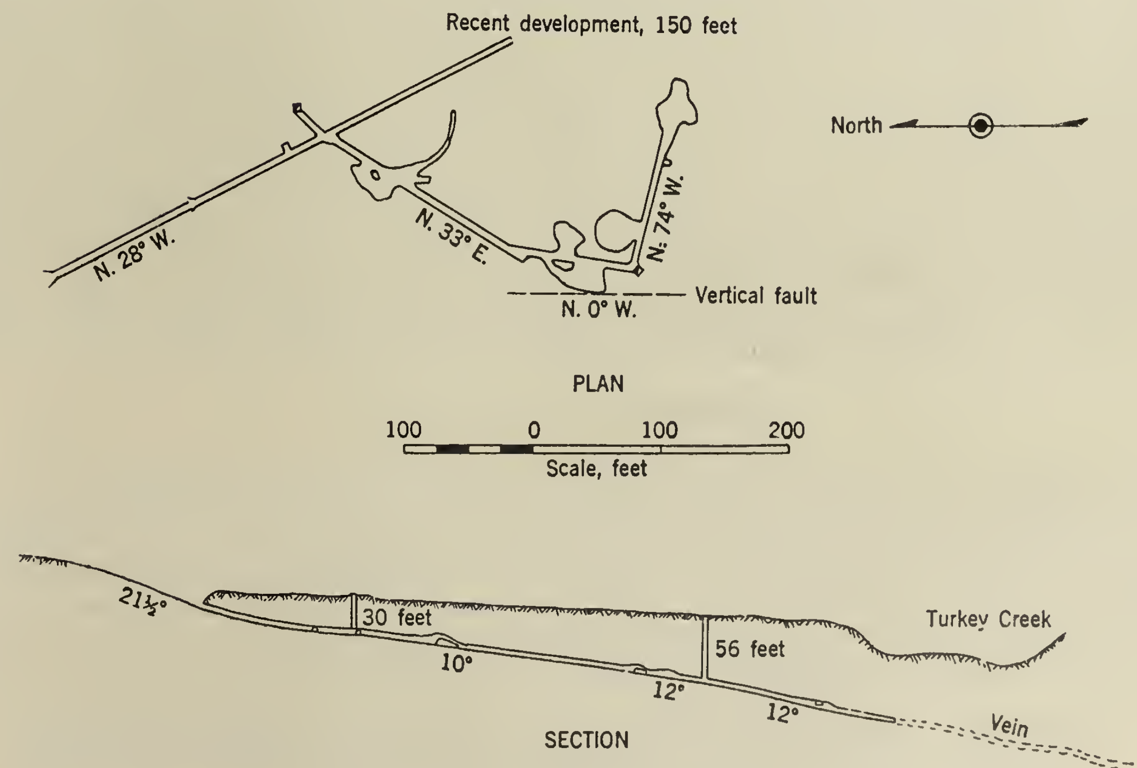

Figure 2.—Plan and section of Golden Belt incline.

The vein is a fissure with a dip varying from 10° to 23°. The strike is about N. 60° E. There is evidence to show that the ore has been deposited in the fissure vein along a thrust fault and in the accompanying shear zone. The fault cuts the schist obliquely, leaving occasional "horses" of country rock in the shear zone. The zone varies in width from 6 inches where it cuts the diabase to 8 feet where it is in the porphyry. The outcrops can be traced on the surface for approximately 1,800 feet.

Figure 1 shows the five unpatented claims, the camp, underground workings, and a dotted sketch of open-cut and surface workings along the outcropping of the vein. According to Lindgren:[3]

> About 2 miles east of Turkey Creek station, in the flat at an altitude of 3,100 feet, are numerous openings on several very flat quartz veins. The country rock is schist, and the veins cut squarely across the schistosity. These veins are said to extend up to Dripping Springs on the north. They are as much as 6 inches wide and are formed of quartz filling with excellent comb structure and a little pyrite and galena. Some of this ore is said to contain $100 in gold to the ton. Much mining has been done on a small scale, and a considerable amount of ore has been shipped. ....

The quartz vein on the Golden Belt property varies in width from a few inches to 3 feet and carries from $5 to $40 in gold per ton and from 1 to 10 ounces of silver. In the upper portions of the mine, above the water level, the vein is highly oxidized and contains the oxides of iron and lead and some free gold. The average value of this ore as determined from several engineers' reports is $10.50 per ton. Below the water level or 50 feet below the surface the ore is almost entirely sulphide and is composed of auriferous pyrite and galena. The galena carries the greater part of the gold. Mill-head assays show the silver to be decreasing as depth is attained. No free gold has been found in the sulphide ore. The schist and porphyry carry finely disseminated pyrite both above and below the vein for a distance of a few inches to several feet. These finely disseminated sulphides carry but little gold or silver; the precious metals seem to be contained in the massive sulphide.

PROSPECTING AND DEVELOPMENT

The outcrops of the vein have been developed by open-cuts, trenches, and adits. A single diamond-drill hole has been drilled 230 feet in advance of the lowest face on the incline. This hole intercepted the "blanket vein" at a depth of 80 to 95 feet and indicated that it was continuous. As the schist is too easily fractured to core well and the vein matter too friable, little core was obtained; however, the cuttings which were saved showed heavy sulphide minerals in the last 3 feet of the hole and a smaller amount of sulphide

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3 Lindgren, Waldemar, Ore Deposits of the Jerome and Bradshaw Mountains Quadrangle, Ariz.: U. S. Geol. Survey Bull. 782, 1926, p. 158.

from the preceding 12 feet. Assays of this sludge were not considered representative, as some of the drilling water was lost and undoubtedly carried away some of the heavier sulphide cuttings. The diamond drilling was done by the company at a cost of approximately $1 per foot; bortz was used as a cutting medium. A used portable Bravo diamond-drilling machine equipped with an Overland gasoline engine was lent to the company.

Until the past few months the development of the vein has consisted of extending the incline, turning indiscriminately to follow the ore but continuing down on the vein. As a result, the incline shaft zigzags its way along the vein for a distance of about 675 feet. On account of the turns and the variation in grade, rollers have been set on the ties, on stulls at the side walls or across the back, to reduce the friction in hoisting and to prolong the life of the cable. The stulls also carry the light and signal-bell wires. The shaft averages 6 by 6 feet in size and is not timbered except at the portal.

Recently an incline has been sunk 150 feet on a dip of 14°, extending the upper part of the shaft in a straight line (fig. 2). The work was done in 1 month, at the rate of one shift per day; drilling and shoveling were done at different times to present a clean face for drilling to prevent increasing the power demand by running the compressor and hoist at the same time. No water was encountered in sinking.

The following tabulation shows the cost of this work. The expense of hoisting equipment, air equipment, tools, and supervision is not included.

Cost of sinking 150 feet of shaft

Labor:	
2 muckers at $74 per month	$148.00
1 miner (foreman) at $124 per month	124.00
1 hoistman at $74 per month	74.00
2 surface men at $74 per month; sharpening steel, hauling supplies, making repairs, etc.	148.00
Power:	
Hoist, air compression, lighting, etc.	100.00
Powder, fuse, and caps	277.50
Supplies:	
Spikes, rails, ties, etc.	50.00
Insurance	11.24
Costs:	
Total for 150 feet of shaft	932.74
Total cost per foot	6.22
Labor	3.29
Explosives	1.85
Power, supplies, etc.	1.08

Detailed costs of all other sinking, drifting, raising, and stoping are not available but are included in the totals in the summary at the end of the paper.

STOPING

No definite system of mining can be decided upon until the flat vein can be traced and the extent of the orebodies determined. There has been no systematic plan of stoping; ore has been stoped only to furnish material to the mill and to prevent suspension of operations when water has delayed progress in the incline. The relatively small amount of ore mined separately from development work has been taken out by an open-stope method. The stopes are very flat and average about 5 feet in height. The ore is shoveled by hand from turn sheets into mine cars; part of it must be shoveled twice. The ground stands very well; 8- by 8-inch stulls are used where needed for support in the stopes. Both in the incline and in the stopes an attempt is made to blast the ore and waste separately so that sorting will be unnecessary. Drilling is done by jackhammers mounted on columns with arms.

Scrapers will probably be used in stoping in the flat vein when the mine is put on regular production.

UNDERGROUND AND SURFACE HAULAGE

A single $\frac{3}{4}$-ton mine car controlled by the hoist is used for all haulage and hoisting. The hoist is a Novo P117 tugger powered by a low-speed 10-hp. motor with a circuit breaker. The hoist drum carries 800 feet of 3/8-inch, 9 by 16 plow-steel cable. The life of the cable is about 4 months and it costs about 10 cents per foot. The haulage incline is about 675 feet in length and the broken rock must be hoisted a vertical distance of about 145 feet. The car is hoisted to the surface at the rate of 4 to 6 trips per hour, requiring about 5 minutes to raise, dump, and return the car to the shovelers. At the hoisthouse the car is switched and lowered by gravity under control of the hoist to the mill crusher bin or to the waste dump. A headframe is not necessary.

VENTILATION AND DRAINAGE

Two raises to the surface (fig. 2) furnish ventilation; one contains a conduit for pipes from the pump and a power line to the pump motor. The first raise cuts the vein and incline at a point 117 feet from the portal and 30 feet below the surface; the second raise, which is 394 feet from the portal, is 56 feet to the surface.

The water level during wet weather was encountered at a depth of about 50 feet below the surface or about 343 feet down the incline from the portal. The fact that the incline dips beneath Turkey Creek may create a serious water problem as depth is attained.

A 6- by 6-inch Gould Pyramid pump with a capacity of 50 gallons per minute and driven by a 5-hp. motor handles all water during the dry season. During wet weather, however, this pump and an American Well Works sinking pump with a rated capacity of 225 gallons per minute and 100 feet lift, driven by a 15-hp. motor, will not handle the flow of water with more than a 50-foot lift. As the bottom of the incline is more than 100 feet below the surface, the larger pump is useless for draining the lower part of the mine.

MILLING PLANT

Early activity on the property was along the line of experimentation in milling rather than of development of an orebody. Amalgamation, cyanidation, and concentration methods were attempted without success. A large part of the mill equipment used in this work was on the property when it was taken over by the present company, so that the cost of rebuilding the plant was not excessive.

The mill is a simple flotation plant of 50 tons maximum daily capacity, installed as a pilot mill to help defray costs of mine development. The operation is intermittent, depending on the mine development output. The ore is very amenable to treatment by flotation methods, and a concentrate is produced by a single rougher operation. For regular operation the milling plant should be augmented by a set of rolls to increase grinding capacity, a cleaner cell, increased ore and concentrate bin capacity, and a thickener for the concentrate pulp before filtration. At present the mill lacks flexibility, which results in intermittent operation and higher operating costs.

The general flow sheet of the mill is presented in figure 3. The building is constructed of galvanized iron and timber, with concrete foundations for the machines. The air-compression equipment for the mine is also housed in the mill building and is tended by the mill operator. All machines are individually motor-driven and require a running load of approximately 47 hp. The following tabulation shows the power requirements of the various units.

Operation	Horsepower load (estimated)	Kilowatt-hours per ton	Percent of total
Breaking	5	1.5	10.6
Grinding	30	18.2	63.9
Classifying and feeding	2	1.2	4.3
Flotation	4	2.4	8.5
Filtering	4	2.4	8.5
Water supply	$1\frac{1}{2}$	.9	3.2
Miscellaneous ...	$\frac{1}{2}$	.3	1.0
Totals	47	26.9	100.0

The above data are based on the treatment of 725 tons of ore in 39 days of operation, the daily operation averaging 15 hours. The breaker is run approximately one half the time of mill operation.

EQUIPMENT AND COSTS

As stated before, some development work had been done and some equipment was on the ground when the present company took over the property. An appraisal has been made of the value of this development work and equipment as of May 1, 1931. The figures are shown in the accompanying tabulation. The cost of new equipment, amount and cost of new development, milling costs, transportation, overhead, and total costs are also shown in the tabulations.

Cost of equipment

Camp:		
Roads	$ 100.00	
Camp buildings:		
Six 3-man bunk houses, 3-room manager's quarters with bath, dining room and kitchen, office, storehouse, and 2 latrines	2,300.00	
Mine buildings:		
Small hoist house, blacksmith shop, and powder house	200.00	
Total roads, camp, and mine buildings	2,600.00	
Camp equipment:		
Furnishings for bunk houses, manager's quarters, office, dining room, kitchen, etc., with electric refrigeration and washing machine and radio	3,507.00	
Total cost of camp		$6,107.00
Mine:		
Blacksmith-shop equipment:		
Blower, anvil, tank, and tools	100.00	
Assaying equipment and supplies:		
Gasoline furnace, balances, etc.	418.34	
Drilling equipment:		
2 jackhammers with air and water hose and fittings, 41-inch column, 54-inch column, and arm and collar	500.00	
Hoisting equipment:		
1 Novo tugger driven by 10-hp., 950-r.p.m. motor with circuit breaker	150.00	
2 ore cars, 1/2- to 3/4-ton capacity	70.00	
800 feet of 3/8-inch, 9 by 16 plow-steel cable (new)	80.00	

Cost of equipment - Continued

Mine (Cont'd):		
Air-compression equipment:		
1 Ingersoll Rand 7 by 6 class ER1 compressor with receiver and fittings	$ 202.00	
Motor for compressor	25.00	
Pumping equipment:		
1 6 by 6 Gould Pyramid pump, chain driven by 5-hp. motor, capacity 25 gal. per min.	226.00	
1 American well works sinker, capacity 225 gal. per min., maximum head 100 feet, mounted on car wheels, driven by 15-hp. motor with starter	632.00	
Miscellaneous	100.00	
Total mine equipment		$2,503.34
Mill:		
Building, including ore bin and concentrate bin ..	2,850.00	
Crushing equipment:		
1, 7- by 10-inch Blake type crusher driven by 10-hp. motor with starter	275.00	
Grizzly, 6 by 3 feet	25.00	
Ore feeder, 30-inch Challenge	100.00	
Ball mill, 5 by 4 feet, with clutch, driven by 30-hp. motor with starter	425.00	
Balls and liners	502.00	
Chain drag classifier, 15 feet by 22 inches, driven by 3-hp. motor which also drives ore feeder and reagent feeder	185.00	
Flotation equipment:		
Single rougher cell	190.00	
Blower:		
Connersville, 285 cubic feet per minute capacity, driven by 10-hp. motor	222.00	
Filter:		
1- by 4-foot Dorrco, including reduction gear, $1\frac{1}{2}$-hp. motor, and vacuum pump with 5-hp. motor	1,125.00	
Water-supply lift pump, V-belt driven by 3-hp. motor	100.00	
Miscellaneous equipment, including reagent feeder and tanks, drive belts, launders, piping, wiring, line shafting and bearings, tools, foundations, etc.	1,688.00	
Total cost of mill equipment		1/ 7,687.00
Transportation:		
Truck	950.00	950.00
Total cost of equipment		17,247.34

1/ Includes $316.35 for repairs of equipment off property.

Cost of operation

Development:		
Labor:		
Open cuts, 750 linear feet (estimated)		
Tunnels and drifts, 598 linear feet (estimated)		
Raises, 95 linear feet		
Incline shaft, 825 linear feet		
Direct labor including $24 per month for board and room, per man	$13,196.15	
Repairs off property	142.95	
Total labor cost		$13,339.10
Supplies:		
Powder, fuse, and caps	1,324.90	
Miscellaneous equipment, tools, and supplies	2,064.84	
Total cost of supplies		3,389.74
Power (mine only)	795.06	795.06
Total development cost		17,523.90
Transportation:		
Supplies and repairs for truck or automobile	998.50	
Truck driver including hauling 258 tons of concentrates	193.50	
Total cost of transportation		1,192.00
Milling:		
Labor for milling 2,370 tons of ore, including mill-construction costs	3,023.88	
Power 1/	1,590.12	
Supplies 2/		
Total milling cost		4,614.00
Overhead:		
Supervision expenses	1,354.81	
Insurance with Industrial Commission	405.41	
Organization expenses (legal)	108.50	
Office expenses	61.50	
Survey of property	144.00	
Miscellaneous expenses	2,390.07	
Total overhead costs		4,464.29

1/ Mill power costs appears excessive because intermittent operation creates a high power demand cost.

2/ Supply costs included with equipment.

Summary of Costs

	Appraised value as taken over May 1, 1931	Disbursements to April 1, 1932	Total
Camps:			
Buildings	$2,000.00	$ 500.00	$2,500.00
Equipment and supplies	2,755.00	752.00	3,507.00
Roads	100.00	-	100.00
Totals	4,855.00	1,252.00	6,107.00
Mines:			
Equipment and supplies	3,362.00	2,531.08	5,893.08
Development:			
Direct labor and repairs.	6,876.00	6,320.15	13,196.15
Repairs off property	-	142.95	142.95
Power	-	795.06	795.06
Totals	10,238.00	9,789.24	20,027.24
Transportation:			
Truck	100.00	850.00	950.00
Labor and supplies	-	1,192.00	1,192.00
Totals	100.00	2,042.00	2,142.00
Mill:			
Buildings and bins	2,800.00	50.00	2,850.00
Equipment and supplies	2,855.00	1,666.15	4,521.15
Repairs off property	-	316.35	316.35
Labor, construction, and milling	-	3,023.88	3,023.88
Power	-	1,590.12	1,590.12
Totals	5,655.00	6,646.50	12,301.50
Overhead	1,100.00	3,364.29	4,464.29
Total costs	21,948.00	23,094.03	45,042.63

Smelter returns

The net smelter returns during the development were as follows:

Crude ore	106 tons	$ 891.49
Concentrates	285 tons	8,523.63
Total		9,415.12

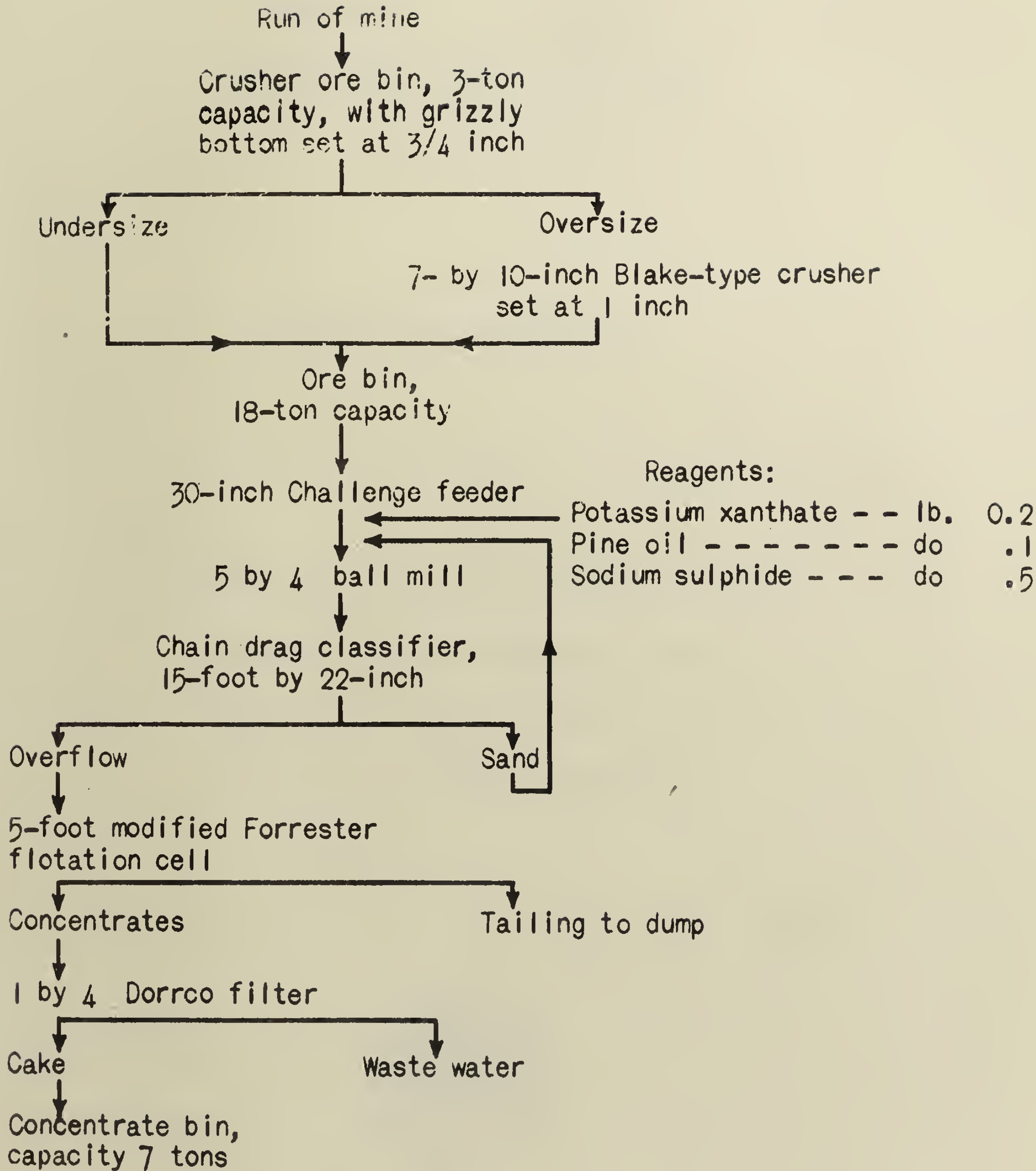

Figure 3.- Flow sheet of mill

I. C. 6736

June, 1933

Department of Commerce

UNITED STATES BUREAU OF MINES

Scott Turner, Director

INFORMATION CIRCULAR

MINING METHODS AND COSTS AT THE TEZIUTLAN COPPER MINE OF THE MEXICAN CORPORATION, S. A., TEZIUTLAN, PUEBLA, MEXICO

BY

Ernest Ph. Herivel

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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING METHODS AND COSTS AT THE TEZIUTLAN COPPER MINE OF THE MEXICAN CORPORATION, S. A., TEZIUTLAN, PUEBLA, MEXICO[1]

By Ernest Ph. Hérivel[2]

LOCATION AND HISTORY

The mine and treatment plant of the Teziutlan unit of the Mexican Corporation, S. A., are located at Aire Libre, in the District of Teziutlan, near the northeast edge of the State of Puebla, Republic of Mexico. The property is leased from the Teziutlan Copper Co.

The town of Teziutlan is the terminus of a branch railroad starting from Oriental Junction on the Mexico-Jalapa-Veracruz narrow-gage system of the National Railways of Mexico. A branch line 18 km in length from Teziutlan to Aire Libre belongs to the Teziutlan Copper Co. and is leased and operated in conjunction with the mine.

Aire Libre is on the Mexican Gulf slope at an altitude of 4,800 feet above sea level, while Teziutlan is about 1,470 feet higher. Because of the heavy grades on the company railway it is necessary to use Shay geared locomotives for hauling supplies and concentrates between these two points.

Mining operations at this property commenced in 1892 upon the discovery of an outcropping body of copper ore of sufficiently high grade to stand the heavy cost of shipping by muleback and railroad to smelters in northern Mexico. Later a smelter with blast furnaces and converters was built near the mine, in which crude ore was smelted. As the mine workings deepened the presence of zinc in the ore began to affect the smelter operations so adversely that a roasting plant was constructed, and still later a mill and concentrating plant were put into operation in an attempt to eliminate the zinc. The first concentration scheme, consisting of a combination of tabling and bulk flotation, was successful for a time; but a continued decrease in the copper content of the ore, accompanied by an increase in zinc and iron contents, finally brought about the abandonment of this scheme in view of the impossibility of producing a concentrate of sufficient grade to smelt profitably.

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6736."

2 One of the consulting engineers, U. S. Bureau of Mines.

The property was closed down for a number of years until the advent of selective flotation when, after a considerable period of pilot-plant research, a treatment scheme was evolved which gave a satisfactory separation of the constituent minerals and produced marketable copper and zinc concentrates.

After the necessary additions had been made to the concentration plant, active operations were resumed in September 1925 and continued until December 1931, when the continued fall in the price of base metals brought about a complete shut-down.

GEOLOGY AND PHYSICAL CHARACTERISTICS OF THE OREBODIES

The orebodies, which are flat-lying lenticular deposits, are found in metamorphosed igneous and sedimentary pre-Cambrian rocks. Mineralization occurred after the period of metamorphism and is believed to be related to a later intrusion of dolerite. Other igneous intrusions are bostonite sills and dikes and an andesite dike.

Four ore bodies have been discovered at successive depths, which are called, respectively, Aurora, Cometa, Volcan, and Minerva. (See fig. 1). The prominent outcrop of the first led to the discovery of the mine; the others were located by diamond drilling. Numerous faults, mostly of pre-mineral age, are responsible for a considerable amount of displacement within the orebodies (fig. 2).

The Aurora and Cometa lenses have mica-schist footwalls and phyllite hanging walls; the Volcan and Minerva lenses have both walls of phyllite. The hanging wall of the Minerva is a soft grey phyllite, in contrast with a fairly hard, dark phyllite over the other bodies.

The ore is a massive sulphide, the principal minerals of which are chalcopyrite, sphalerite, pyrite, and galena, in a gangue of quartz and, occasionally, mica schist. There is some mineralization of the mica-schist footwall in certain sections of the Aurora and Cometa lenses, but in general the mineralization ends sharply at the contact of ore and wall rock. The lenses have a general east-west strike and dip toward the south. The dip is irregular, probably conforming with folds in the encasing rocks, and ranges from less than 20° to 65° in a few isolated cases; the average is approximately 35°.

The following tabulation gives the approximate size of the orebodies, excepting the Minerva, which is only partly developed:

Approximate size of orebodies (Minerva partly developed)

	Length		Height		Width	
	Meters	Feet	Meters	Feet	Meters	Feet
Aurora	370	1,210	270	890	3.6	11.8
Cometa	390	1,280	280	920	2.2	7.2
Volcan	230	750	140	460	2.2	7.2
Minerva	270	890	100	330	1.7	5.6

GENERAL STATISTICS

The normal daily tonnage during the last period of operations has been 220 tons per working day produced from the mine, and 180 tons daily milled.

The average grade of the ore is tabulated below in order to compare the whole last operating period with the year ending June 1931 and with the month of December 1931. The increase in zinc and lead is due to the production from the Minerva orebody, which started in January 1931. By the end of the year about 90 percent of the total tonnage mined was from the Minerva, the other orebodies being practically worked out.

	Gold, pennyweight	Silver, ounce	Copper, percent	Zinc, percent	Lead, percent
September 1925 to June 1931	0.585	2.677	3.27	11.92	1.20
July 1930 to June 1931	.685	3.159	3.20	13.82	1.72
Month of December 1931	.806	3.215	3.02	13.99	2.15

There are no data available regarding total production from earlier operations, but the following figures give this information for the years 1929 and 1930, and from September 1925 to the end of 1930.

	1929 and 1930	September 1925 to December 1930
Ore produced........................ tons	129,233	325,281
Copper in concentrates produced.....pounds	6,853,000	17,876,345

MINING

The mine is worked through an adit on the 160-meter level that was driven below the Aurora orebody and into the Cometa. When the workings in the Cometa extended below the adit level, an interior three-compartment shaft was sunk, from which the 190-meter and 220-meter levels were developed. Later this shaft was sunk to the 250-meter level where a crosscut was driven to the Volcan orebody. The exploitation of the lower levels of this orebody made necessary another smaller shaft to the 300-meter level. The discovery of the Minerva orebody, deeper and farther from the adit entrance, brought up more serious handling and ventilation problems and it was decided to sink a shaft from the surface about 1 mile from the tunnel entrance.

SHAFT

The Minerva shaft was sunk in 1930. It has three compartments, each 5 by 6 feet in the clear, timbered with 8- by 8-inch oak sets at 6-foot intervals and lagged with 2- by 12-inch oak plank. The guides are of treated pine, 4 by 8 inches in section. In two compartments single-deck cages are used for hoisting; the third is divided into two smaller compartments, one 2 1/2 by 5 feet for pipes and electric cables, the other for a ladderway.

Before starting shaft work the 160-meter, 250 meter, and 300-meter levels were driven to the shaft location, and from each level a pilot raise was put up; sinking from the surface commenced at the same time. After the pilot raises were broken through they were stripped to full size and timbered to the 300-meter level; from there the shaft was sunk full size for a further distance of 82 meters and the 370-meter level was opened, leaving a 12-meter sump. Fifty-pound jackhammers were used in sinking operations, and stoping drills were used in the pilot raises.

WINZES

Very little development is done by winzes, only two having been started, shortly before cessation of operations, to explore the Minerva orebody below the 370-meter level. These winzes are 5 by 8 feet in section, and are divided so as to leave the hoisting way 5 by 5 feet and the remaining space for ladders. They are sunk on the ore and the only timber required is for stulls and lagging between ladders and ore-pass. Hoisting is done with small tugger hoists, using round buckets of about 250-pound capacity running on timber skidways.

RAISES

Raises likewise are usually 5 by 8 feet in section, with a 5- by 5-foot chute at one end; 8- by 8-inch stulls and 2- by 12-inch plank lagging are used for separating manway and chutes. When the adjacent ground is stoped, additional stulls are placed at both ends and lagged on the outside with 3-inch poles.

DRIFTS AND CROSSCUTS

Drifts and crosscuts are driven 5 by 7 feet in the clear. Only about 10 percent of the workings in country rock require timbering, but in the drift on the Minerva orebody about 50 percent is timbered because of the softer hanging wall. Sets of 10- by 10-inch oak are used with 3-inch pole lagging overhead. Drilling is done with 180-pound drifting machines, using two to a face. In average ground a standard 17-hole round is drilled. All breaking is done with 40 percent strength gelatin dynamite in 1- by 8-inch sticks. Wherever possible electric blasting is used in development work.

STOPING

The room and pillar method has been used generally in the first three orebodies. About 15 percent of the ore was left as pillars, spaced according to height of stope and condition of hanging wall. A section of the orebody, usually the length between fault slips or offsets, was worked out from a main level or sublevel to the next above, or to the top of the ore lens, after which pillars were mined, retreating from the outer or upper extremities, and allowing the back to cave in. If the pillars had been standing for a long time it was found advisable to use some additional form of support to prevent the roof from caving too fast as the pillars were removed. Between levels this was most easily accomplished by dry-walling and pack-filling sections of the stopes; where waste for filling was not available rows of stulls had to be put in before removing each successive row of pillars.

Main haulage levels are generally spaced at 30-meter (100-foot) intervals, but where the lenses flattened out, sublevels were put in to facilitate handling of ore from stopes. In some cases the ore was dumped through chutes to a main level below, and in others the cars were hauled up an incline to the level above by means of small hoists. Stope chutes were put in as convenient between pillars and the ore moved to these by shovel and wheelbarrow. Raises were run between levels only as required for roadways and ventilation.

The cut-and-fill method was adopted in the Minerva stopes because of the soft hanging wall. With this method raises are put up at 30-meter (100-foot) intervals along the level, and from these the first stope floor is broken by stope drifts between the raises, leaving a 5-meter (16-foot) pillar above the level. Alternate raises are run up to determine the upper limits of the ore, which has been found to extend 20 meters (66 feet) to 50 meters (164 feet) above the level, measured on the plane of orebody. The other raises are carried up as the stope advances and serve as ore passes. For purposes of ventilation, one raise has been driven to a connection with workings on the 300-meter level.

After the stope drifts are connected, they are stripped to the full width of the ore, and the back in a section between a raise and chute is broken down to a height of 12 feet. As soon as the broken ore is removed the section is filled with waste to a height of 7 feet. Each successive cut is 7 feet high, and is immediately followed by filling.

Where stopes are sufficiently straight, the ore is handled by scrapers set up at the chutes midway between the raises, and pulling alternately from either side, so that while one section is being stoped the other is being filled. Where the orebody is folded and the stope sections consequently crooked, wheelbarrows are used; in such cases the lower parts of the waste raises are used as additional ore chutes in order to shorten the distance that the ore has to be moved in the stopes. The filling is handled in the stopes by shovel and wheelbarrow. In the higher sections of stopes, scrapers are used to pull the ore down the chutes, as the dip is too flat to permit it to flow by gravity.

Waste for filling is obtained by breaking caving stations at the tops of raises above the orebody. When lower levels are worked the waste can be trammed directly to chutes for delivery to the workings below, and waste from development work also will be disposed of in this manner.

Very little timber is used for support in stopes. Beyond an occasional stull to hold up a slabby portion of the hanging wall, the consumption of timber in stoping is mostly for construction of chutes and for raise timbering.

The ore is broken by inclined back holes, using stoping drills. All drills not of the wet type are equipped with spray attachments to allay the dust.

UNDERGROUND TRANSPORTATION

End-dumping cars of 20-cubic-foot capacity are used for transportation of both ore and waste. From stopes to shafts the tramming is done by hand. The loaded cars are hoisted in cages to the main tunnel, where trains are made up and hauled to the mill bins or waste dumps by 4 1/2-ton electric locomotives of the overhead-trolley type. Before being dumped into the mill storage bins the ore is weighed on track scales. The mine track is of 20-inch gage; 16-pound rails are used on all levels except the main tunnel, where 45-pound rails discarded from the railway have been put into service.

COMPRESSED AIR, VENTILATION, DRAINAGE, AND SAFETY

The compressor plant, consisting of four compressors with a total rated capacity of 3,200 cubic feet of free air per minute, is located at the entrance to the main tunnel. The air is delivered to the mine through a 6-inch pipe line, from which 4-inch lines are run off on the levels; further subdivision to stopes and working faces is through pipes of different smaller sizes, according to the number of machines served. The blacksmith and drill-sharpening shop is near the compressor house. Leyner drill sharpeners are used.

Natural ventilation through surface raises was sufficient for working the Aurora and Cometa orebodies. To remove the vitiated air and smoke from the Volcan workings, 11-inch rigid tubing was installed along the 250-meter level; exhaust fans were installed where needed, one being placed midway along the tubing as a booster. Propeller fans with stationary guide vanes were used in development headings. After connections were made with the Minerva shaft, natural ventilation supplied all needs except in the case of long drifts. The mine is naturally cool, and the temperature rarely exceeds 70°F.

Drainage and pumping are important at this mine. To handle water from the lower Cometa and upper Volcan workings, a three-stage centrifugal pump is installed at the 250-meter level and pumps to the tunnel level. It is driven by 75-hp. motor and has a capacity of 800 gallons per minute for a 300-foot lift. Water from the lower section of the Volcan orebody is collected at the 300-meter level and pumped to the 250-meter level by a 300-gallon, 125-foot lift, centrifugal pump driven by a 25-hp. motor.

A greater flow of water was naturally expected from the deeper workings, and a 4-stage centrifugal pump of 1,000 gallons per minute capacity, 800-foot lift, driven by a 300-hp. motor, was installed at the 370-meter level to pump directly to the tunnel level.

There is a considerable seasonal variation in the flow of mine water; surface infiltrations cause a marked increase, even in the lowest workings, soon after the commencement of the rainy season. The average flow of mine water varies from 650 gallons per minute during the months of March to May to 1,500 gallons per minute in September and October, fluctuating between the high and low points during the remainder of the year.

Three five-man crews are trained in the use of oxygen-breathing apparatus for rescue work, besides which weekly classes are given in first-aid and safety measures. First-aid kits are kept at easily accessible points throughout the mine, and gas masks are kept at different places in the lower levels, where they can be quickly found in case of emergency. The use of protective or "hard-boiled" hats is compulsory underground. The company maintains a well-equipped hospital, with doctor and nurse in attendance, for the care of injured workmen.

POWER

Power is generated at two company-owned hydroelectric plants, with a capacity of 2,430 kilovolt-amperes, situated about 7 kilometers from the mine. Sixty-cycle alternating current is generated and delivered to the property at 5,000 volts; transformer stations are located at the tunnel entrance for compressors, haulage, and mill service; and at the Minerva shaft for hoists and pumping. All motors are 440-volt, but for lighting and shaft signals the power is stepped down to 110 volts in small transformers.

COSTS

The following tables give detailed costs of underground operations, both in dollars (United States) and in units of man-hours and supplies; they cover the period from July 1930 to June 1931. The tonnage is that actually milled, as all costs are calculated on this basis. The differences between tonnage mined and that milled is negligible in a period of 12 months. Weights are in dry tons of 2,000 pounds.

Table 1.--Summary of costs

Tons of ore mined during period: 65,764 Period: July 1930 to June 1931, inclusive

Mining methods: Open stope and pillar, cut and fill

Underground costs per ton of ore mined

	Labor	Supervision	Compressed air, drills, and steel	Power	Explosives	Timber	Other supplies	Total
Development	$0.458	$0.025	[1] $0.135	$0.008	$0.119	$0.068	$0.113	$0.926
Stoping	.877	.041	.252	......	.216	.107	.078	1.571
Transportation ..	.209	......	....	.008	....	....	.052	.269
General underground expense .	.152	.070	....	[2] .070	....	....	.020	.312
Surface expense (directly applicable to underground operation)	.080	......	....	......	....	....	.030	.160
Total	1.776	0.136	0.387	0.086	0.335	0.175	0.343	3.238

1 Includes power for air compression.
2 Principally power for pumping.

Table 2.--Detail of costs in units of labor, power, and supplies

Tons of ore mined: 65,764
Mining methods: Open stope and pillar, cut and fill
Period: July 1930 to June 1931, inclusive

	Development	Mining	Total
Labor (man-hours per ton):			
Breaking (drilling and blasting)...	0.449	1.068	1.517
Timbering........................	.104	.386	.490
Filling..........................	.....	1.056	1.056
Shoveling........................	.631	1.788	2.419
Haulage and hoisting..............	.562	3.097	3.659
Supervision.......................	.085	.293	.378
Pumpmen and mechanics..............	.....	.....	1.054
Storekeepers and tool changers.....	.....	.....	.267
Miscellaneous underground..........	.....	.....	.114
Total labor underground............	.....	.....	10.954
Surface labor directly chargeable to underground operations.........	.....	.....	1.027
Average tons per man per shift.....	.....	.....	.674
Labor, percentage of total cost.......	58.2	71.2	69.4
Power and supplies:			
Explosives, 40 percent gelatin dynamite, pounds per ton..........	0.522	0.947	1.469
Timber, board feet per ton.........	2.450	5.547	7.997
Total power, kilowatt-hours per ton	.....	.....	62.138
(1) Air compression.............	9.072	21.127	30.199
(2) Hoisting..................	1.864	2.237	4.101
(3) Tramming..................	.373	.497	.870
(4) Pumping...................	.....	.....	25.042
(5) Ventilation...............	.375	.495	.870
(6) Lighting..................	.....	.....	1.056
Other supplies, in percentage of total power and supplies.........	42.4	29.3	31.1
Power and supplies, percentage of total cost......................	41.8	28.8	30.6
Percentage of total mining cost....	28.6	71.4	.....

Table 3.--Detail of development costs in units of labor, power and supplies
Period: July 1930 to June 1931, inclusive

	Sinking		Drifting and crosscutting	Raising	Total, all development
	Shaft	Winzes			
Size of excavation, feet	8 by 20	5 by 8	6 by 8	5 by 8	....
Timbered or not	Yes....	Yes...	About 30 percent	Yes....	....
Physical properties of rock:					
(a) Hard or soft	Hard...	Hard..	Hard....	Hard...	....
(b) Firm or loose	Firm...	Firm..	Firm....	Firm...	....
(c) Swelling or running ground		Occasionally slabby			....
Advance, feet	193	46	1,881	862	2,982
Labor (man-hours per foot):					
Breaking	27.254	8.101	7.727	10.880	9.908
Timbering	14.850	1.514	1.444	1.399	2.300
Shoveling	35.465	15.482	14.767	7.172	13.922
Haulage and hoisting	9.784	12.757	12.168	13.470	12.400
Supervision	1.873	1.873	1.873	1.873	1.873
Total labor	89.226	39.727	37.979	34.794	40.403
Feet per 8-hour shift	.825	.590	1.269	.921	1.092
Power and supplies (per foot):					
Explosives, pounds	43.870	10.978	10.968	5.484	11.512
Timber, board feet	422.794	27.304	29.596	26.212	54.032
Total power, kilowatt-hours	628.585	251.433	251.435	188.578	257.679
(1) Air compression	498.468	195.363	190.365	144.506	197.130
(2) Hoisting	110.002	48.024	39.978	29.984	41.746
(3) Haulage	20.115	8.046	13.046	8.054	11.984
(4) Ventilation	....	....	8.046	6.034	6.819
Other supplies (in percentage of total power and supplies)	28.5	54.7	44.6	53.7	42.4
Labor (percentage of total cost)	57.6	60.5	57.6	60.2	58.2
Power and supplies (percentage of total cost)	42.4	39.5	42.4	39.8	41.8

Table 4.--Detail of development cost in dollars per foot

Period: July 1930 to June 1931, inclusive

	Sinking		Drifting and crosscutting	Raising	Total, all development
	Shaft	Winzes			
Advance, feet..............................	193	46	1,881	862	2,982
Labor (cost per foot):					
Drilling and blasting....................	$13.495	$ 2.540	$ 1.769	$ 2.245	$ 2.822
Timbering................................	7.353	.472	.329	.260	.766
Shoveling................................	17.561	4.852	3.388	1.341	3.736
Haulage and hoisting.....................	4.845	3.996	2.791	2.708	2.794
Supervision..............................	.551	.551	.551	.551	.551
Total underground labor..................	43.805	12.411	8.828	7.105	10.649
Surface labor charges....................	1.235	1.235	1.235	1.235	1.235
Total development labor..................	45.040	13.646	10.063	8.340	11.884
Power and supplies (per foot):					
Explosives...............................	$10.003	$ 2.503	$ 2.501	$ 1.250	$ 2.625
Timber...................................	11.737	.758	.822	.728	1.500
Total power..............................	1.923	.769	.769	.577	.788
(1) Air compression......................	1.525	.598	.583	.442	.604
(2) Hoisting.............................	.337	.147	.122	.092	.127
(3) Haulage..............................	.061	.024	.040	.025	.037
(4) Ventilation..........................	....	....	.024	.018	.020
Other supplies...........................	9.448	4.870	3.301	2.960	3.624
Total power and supplies.................	33.111	8.900	7.393	5.515	8.537
Total labor, power, and supplies (per foot)..	$ 78.151	$ 22.546	$ 17.456	$ 13.855	$ 20.421
Labor (percentage of total cost)............	57.6	60.5	57.6	60.2	58.2
Power and supplies (percentage of total cost)	42.4	39.5	42.4	39.8	41.8

Table 5.--Detail of stoping costs in dollars per ton

Mining methods: Open stope and pillar, cut and fill

Period: July 1930 to June 1931, inclusive

Tons of ore mined: 65,764

	Cost per ton	
Ore breaking		$ 1.321
Labor	$ 0.802	
Explosives	.216	
Supplies	.051	
Compressed air	.086	
Steel and tool sharpening	.083	
Machine repairs	.083	
Timbering		.212
Labor	.070	
Timber	.107	
Supplies	.019	
Tool sharpening	.004	
Framing and shop charges	.012	
Tramming		.164
Labor	.111	
Supplies	.017	
Shop charges	.012	
Car repairs	.022	
Power	.002	
Hoisting		.105
Labor	.074	
Supplies	.011	
Shop charges	.014	
Power	.006	
Sampling		.015
Labor	.008	
Supplies	.001	
Assaying	.006	
Pumping		.192
Labor	.108	
Supplies	.016	
Power	.068	
Miscellaneous labor charges		.033
Miscellaneous supplies		.003
Miscellaneous power		.002
Supervision		.063
Engineering		.048
Hospital expenses		.066
Housing expenses		.088
Total charges to mining		2.312

MILLING

Capacity of Plant, and Method of Treatment

The crushing and grinding sections of the plant have a normal daily capacity of 500 tons, but the daily capacity of the flotation section is only 200 tons.

Treatment is entirely by the selective flotation method, based principally on the depressing effect of cyanide on sphalerite and pyrite. The higher galena content of Minerva ore led to a series of mill tests to determine the advisability of floating this mineral as a separate product. While this is still in an experimental stage, some lead concentrate has been produced and shipped.

As the local smelter is not adaptable to their treatment, the lead and copper concentrates are shipped to custom smelters and the zinc concentrate to Europe.

Following is a brief summary of milling and treatment practice.

Coarse Crushing

Ore is delivered from the mine to two circular, steel storage bins, 18 feet in diameter and 30 feet deep, each having a capacity of 500 tons. Through side doors near the bottom of each bin the ore passes by gravity to a 15- by 24-inch Blake jaw crusher, by which it is reduced to 4-inch size. This product is carried by a 26-inch inclined conveyor belt to the top of the mill building proper, where it is dumped directly onto inclined grizzlies, with bars spaced 1 1/2 inches apart. The grizzly oversize goes to two McCully No. 4 gyratory crushers which reduce the ore to a maximum size of 1 1/2 inches. The discharge from these crushers drops directly into the rolls storage bin, which is of wooden construction and has a capacity of 1,000 tons.

Secondary Crushing

The ore is next reduced to minus 1/4-inch size by two 42- by 16-inch Traylor rolls, working in series and crushing dry. After passing through trommels from which oversize is returned to the rolls, the ore is carried by a horizontal tripper conveyor belt, 18 inches wide, to six circular, steel bins, each 8 feet in diameter and 28 feet deep, with a total capacity of 400 tons; these form the storage for the Hardinge mill feed. An automatic sampler cuts samples for mill-heads assay and moisture determination.

Grinding

The grinding plant consists of one 8-foot by 30-inch Hardinge mill and three 8-foot by 22-inch Hardinge mills. Steel balls are used as grinding media, the larger mill being charged with 7 tons of 2-inch balls and the small mills with 5.5 tons of 1 1/2-inch balls. The ball consumption averages 3.75 pounds per ton of ore. The small mills are in closed circuit with model "B"

Dorr duplex classifiers, and the large mill is in circuit with a Dorr bowl classifier, 4.5 feet by 18 feet with 6-foot bowl. The final discharge from this section is 95 percent minus 200 mesh.

Flotation

After thickening and conditioning, the pulp passes to the lead and copper section of the flotation plant. The lead roughing is done in Fagergren machines and the cleaning in Callow pneumatic cells; copper roughing is done in Kraut and Fagergren machines and the cleaning in Callow cells. Tailings from this section are pumped to a zinc conditioning tank, and from there flow by gravity to the zinc section where the roughing is done in Callow cells and the cleaning in McIntosh pneumatic machines.

Filtering

The concentrates are dewatered in Dorr thickeners of concrete construction, the thickened concentrate being removed from the tanks by Dorrco pumps, and filtered in Oliver filters. From the filters the concentrates are transported to storage bins for loading and shipment.

Costs

The following table gives detailed milling costs for the 1-year period from July 1930 to June 1931.

Table 6.--Detail of milling costs in dollars per ton

Tons of ore treated: 65,764
Period: July 1930 to June 1931, inclusive

	Cost per ton	
Coarse crushing		$0.055
Labor	$0.043	
Supplies	.008	
Power (1.445 kilowatt-hours)	.004	
Secondary crushing		.090
Labor	.046	
Supplies	.038	
Power (2.136 kilowatt-hours	.006	
Grinding		.317
Labor	.100	
Supplies	.156	
Power (22.052 kilowatt-hours)	.061	
Thickening and emulsification		.058
Labor	.043	
Supplies	.007	
Power (2.827 kilowatt-hours)	.008	
Flotation		.567
Labor	.155	
Supplies	.067	
Oil and reagents	.274	
Power (25.758 kilowatt-hours)	.071	
Filtering and storing concentrates		.119
Labor	.084	
Supplies	.014	
Power (7.602 kilowatt-hours)	.021	
Weighing, sampling, and assaying		.065
Labor	.027	
Supplies	.005	
Assaying	.033	
Disposal of tailings		.021
Labor	.013	
Supplies	.008	
Laboratory and experimental work		.031
Labor	.017	
Supplies	.001	
Power (0.063 kilowatt-hour)	.001	
Assaying	.012	
General mill expenses		.088
Labor	.069	
Supplies	.013	
Power (0.942 kilowatt-hour)	.003	
Miscellaneous	.003	
Supervision		.050
Royalties		.022
Hospital expense		.025
Housing expense		.044
Total charges to milling		1.552

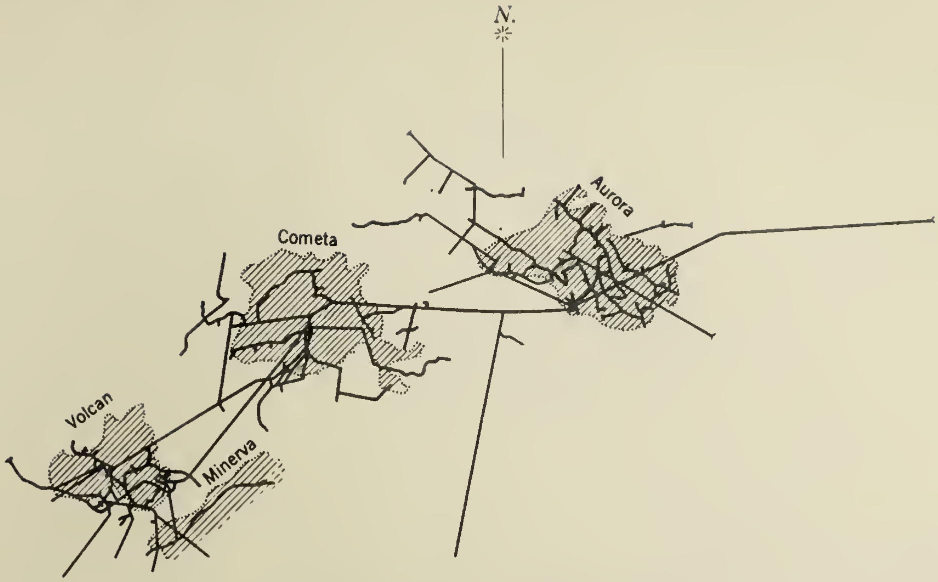

Figure 1.— Mine plan, showing position of orebodies.

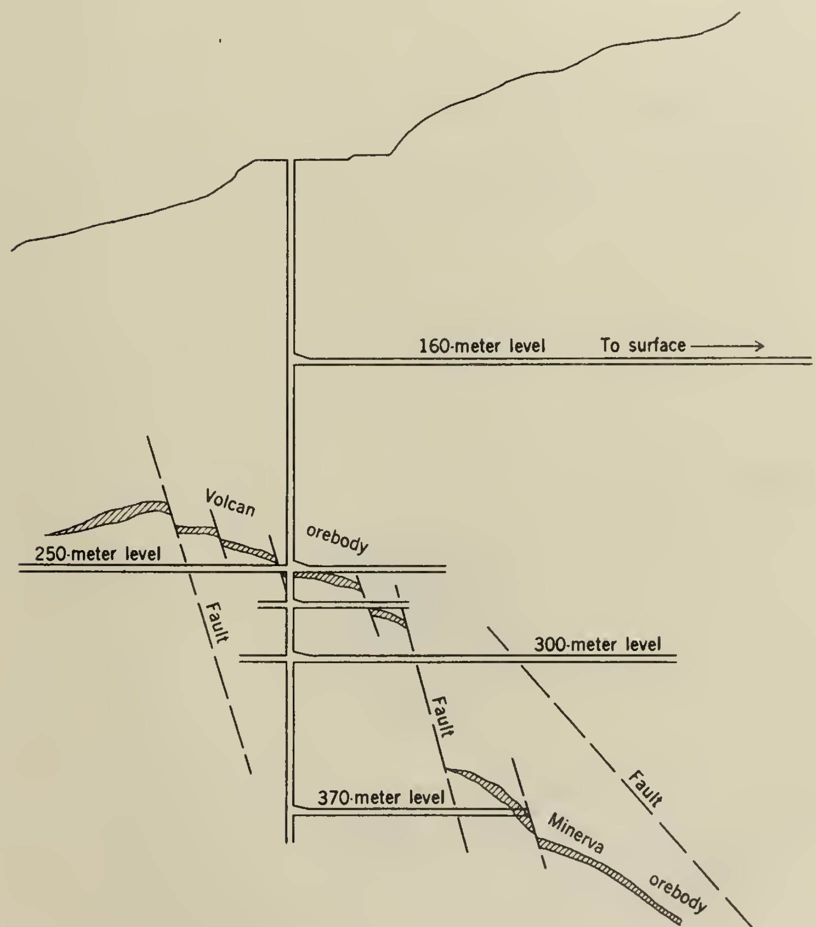

Figure 2.— Section through Minerva shaft, looking N. 54° E.

Petroleum 604

I. C. 6737 July, 1933

Department of Commerce

United States Bureau of Mines
Scott Turner, Director

INFORMATION CIRCULAR

PETROLEUM AND NATURAL-GAS STUDIES OF THE UNITED STATES BUREAU OF MINES

BY

H. C. Fowler

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

PETROLEUM AND NATURAL-GAS STUDIES OF THE UNITED STATES BUREAU OF MINES[1]

By H. C. Fowler[2]

The social economy, or public interest, and national defense require wise development and efficient use of the oil and gas resources of the United States. In this development and use, and in the prevention of economic and physical waste of oil and gas, the objectives of the petroleum and natural-gas industries, the Government (Federal, State, and local), and the public are the same. Differences of viewpoint generally have originated in the failure of one of the triad to have a clear or complete understanding of the motives behind the activities of the other component groups or full knowledge of what each is trying to accomplish.

The United States Bureau of Mines through its petroleum and natural-gas division makes technologic and scientific investigations pertaining to recovery,[3] preparation, refining, and use of oil and natural gas with the view of reducing waste, increasing safety and efficiency, and aiding economic development.

Frequently questions are asked regarding the purpose and scope of studies made and the results obtained by the Bureau of Mines on technical problems pertaining to petroleum and natural gas. In order that there may be available authentic information as to methods whereby the Bureau of Mines is functioning to the benefit of the oil and gas industries, the Government, and the public, and to the better coordination of thought on the part of these groups, this paper has been written. For a clearer understanding, a brief review is first given of the Bureau's earlier work on oil and gas, to show the types of problems studied and the research procedure that has been followed; also, there are indicated the changes in the scope of activities of the petroleum and natural-gas division that have taken place within the last 18 years concurrent with a magnitude of development of the oil and gas industries that finds but few equals. In the latter part of the paper close scrutiny is given to present specific studies on oil and gas and the results that are being obtained.

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1 The Bureau of Mines will welcome the reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6737."

2 Acting chief engineer, petroleum and natural-gas division, U. S. Bureau of Mines.

3 The meaning here conveyed is in accordance with somewhat common usage in the petroleum and natural gas industries - namely, the withdrawal of oil and gas from underground reservoirs by various production methods - rather than in accordance with the more exact definition of regaining that which has been lost.

The Bureau of Mines Function Pertaining to Petroleum and Natural Gas

It has been a motivating thought within the Bureau of Mines since its petroleum division became an officially recognized unit of the Bureau on July 1, 1914, that the chief function of that division is to develop and carry out a coordinated system of research projects pertaining to oil and gas. In performing this function, the best available knowledge and information regarding physical phenomena and other conditions are gathered together, and analyses are made of the data that help to explain the working out of definite laws and fundamental relationships. By the practical application of these relationships to oil and gas fields, pipe lines, and refineries, it is possible to produce oil and gas more rationally, transport them more efficiently, and manufacture them into useful products with less cost.

The Bureau has long recognized that there is a merging from fundamental research in the laboratory to so-called "practical" research. Therefore, in developing the philosophy of any research project it is not enough to establish certain fundamental relationships by laboratory experiment: to be effective the work must have practical application in the field or at the plant.

Through its publications, letters, formal papers, personal discussions, and other means of disseminating information, the Bureau also is slowly yet definitely helping to guide the public mind away from erroneous ideas (evolved during the early life of the petroleum industry because of the fluid and mobile characteristics of oil and gas) to those on a more rational basis, that tend to conform more nearly with the unalterable laws of nature.

Early Work on Oil and Gas

It is significant that many of the early pronouncements and methods of the petroleum engineers of the Bureau of Mines are only now being recognized generally throughout the industry and by the public. In reference to what should constitute adequate legislation for the protection of oil and natural-gas resources, for example, the Bureau of Mines, in 1915, after meetings were held with various State agencies, made the following statement:[4]

> The laws not only protect against waste but also insure a market for natural gas, and thus induce producers to conserve gas instead of allowing it to escape, a rateable marketing of all oil and natural gas offered for sale being provided for. In case production becomes too large for the available transportation and marketing facilities, the transportation facilities must be increased or the oil and gas must be confined until they can be utilized. This provision will prevent large quantities of oil and natural gas from being brought to the surface and stored with a resulting waste of gas and a lowering of oil prices.

4 Fifth Annual Report of the Director of the Bureau of Mines to the Secretary of the Interior for the fiscal year ended June 30, 1915, p. 80.

The foregoing statement reflects what many students of oil and gas conservation believe is the essence of a rational conservation law today.

Another example of a forward-looking viewpoint pertaining to petroleum technology may be found in a series of experiments performed by the Bureau of Mines more than 10 years ago, giving a visible demonstration of the migratory tendencies of gas, oil, and water in the reservoir. The experiments[5] were not carried out in their entirety, but the technique was developed and certain fundamental relationships established which have had wide and definite application, particularly since the water conditions in the East Texas field and elsewhere have become such an important problem to operators, to regulatory bodies, and to the courts of law.

During the first year of its organization (1910-1911), the Bureau of Mines analyzed samples of petroleum (continuing work started by Allen and his associates in the technologic branch of the United States Geological Survey before the Bureau was created) in its fuel-testing laboratory at Pittsburgh, Pa., and a small laboratory was maintained in the Custom House at San Francisco for the purpose of analyzing fuel oils used by the Government. Also, a method of analyzing natural gas (not fundamentally different from the method in general use today) was developed by Burrell and others.[6] Work on various oil and gas problems was continued during the second fiscal year, and during the third fiscal year nearly 150 samples of natural gas were collected and analyzed in a study of the composition of natural gas in different parts of the country. Possibilities of extracting gasoline from natural gas also were studied. Realizing that wastes of oil and gas were occurring on public lands and lands adjacent thereto, some field studies were made, particularly in California, with reference to excluding water from wells by the use of cement.

The need of specifications for petroleum products used by the Government is indicated in the early correspondence of the Bureau of Mines. Joseph A. Holmes, first director of the Bureau of Mines, wrote to a large number of scientific and technical men under date of July 15, 1912, pointing out the wide difference of opinion among manufacturers and consumers of petroleum products in regard to proper specifications. Subsequently, Dr. Holmes suggested an attempt to coordinate and organize "inquiries of national scope through which knowledge in the possession of different investigators could be made available for the general good." A series of conferences, held at Pittsburgh, Pa., in August and September, 1913, was attended by representatives of 34 organizations. Many problems of the petroleum industry, in addition to specifications for petroleum products, were discussed.

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5 Mills, R. Van A., Experimental Studies of Subsurface Relationships in Oil and Gas Fields: Econ. Geol., vol. 15, 1920, pp. 398-421; Relations of Texture and Bedding to the Movements of Oil and Water Through Sands: Econ. Geol., vol. 16, 1921, pp. 124-141.

6 Later reported in Bulletin 197, Bureau of Mines; latest edition published in 1926.

As an outgrowth of those conferences, plans were considered seriously for the organization of what was termed "The American Petroleum Society,"[7] but probably the greatest general good that came out of those meetings was the realization, definitely stated, that "the industry is groping in the dark and that little accurate or even general information is available."

In 1914, the capital invested in petroleum in the United States was said to be about one-half billion dollars, and wastes in drilling, development, storage, and transportation were estimated to be at least $50,000,000 per annum, a large part of which was preventable. Recognizing the condition, the Bureau of Mines started an active conservation campaign, especially stressing the need for the prevention of gas wastage. Corrective measures recommended by the Bureau led, in a short time, to the saving of 350,000,000 cubic feet per day of actually measured open-flow waste of gas. A study also was begun of the decline of oil wells, which later developed into one of the major, if not epic, contributions to the advancement of petroleum technology, and methods were suggested for prolonging the producing life of wells. In transportation, work was directed to efficient methods of construction and operation of pipe lines, cost studies of such methods and operation, and to various types of tank and reservoir construction.

The publication of a report on the physical and chemical properties of petroleums of California[8] was the beginning of a general study of various oils to determine their relative values as sources of commercial products. These crude-oil surveys now include samples from nearly all of the important fields in the Western Hemisphere and from many of the more important fields in the Eastern Hemisphere.

Because very little scientific knowledge of chemistry involved in refining petroleums was available, and most refiners were dependent upon knowledge gained by their apprenticeship to older refiners, studies of refinery technology were started by the Bureau with the view of developing more efficient methods for the preparation of petroleum products.

In those early formative years of the petroleum division of the Bureau of Mines, the whole broad field of engineering and scientific research confronted the small group of technologists who first constituted this division. The insight with which an urgent need was sensed for broad and fundamental studies pertaining to petroleum and allied substances is reflected in the titles and subject matter of many of the earlier publications of the Bureau of Mines, many of which remain today standard references on the subject of petroleum technology.

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7 Explained in detail in Technical Paper 72, Bureau of Mines, 1914. (Out of print.)

8 Allen, I. C., and Jacobs, W. A., Physical and Chemical Properties of the Petroleums of the San Joaquin Valley, Calif.: Bull. 19, Bureau of Mines, 1911, 60 pp.

During the first years of the petroleum division of the Bureau of Mines, there were few petroleum technologists in the employ of the oil companies, and in addition to their basic studies, engineers of the Bureau were called upon to render all manner of service to the companies, frequently taking direct charge of the work, with the consent of the operators, in an effort to bring about corrective measures or improved methods. For example, the Bureau was the pioneer in advocating uniform casing programs and the cementing of wells, and by work on the derrick floor, the use of peg models, cross sections, and other devices, operators were gradually convinced of the harmful effects of drilling into oil-producing formations without adequate safeguards against water encroachment.

It was during this period, when "in the consensus of opinion of well-informed authorities" demand for petroleum likely would soon exceed the productive capacity of the then discovered reserves, that the Bureau of Mines gave to the industry the basic and studied report of Lewis,[9] which laid the ground work for later research and extensive practical application of methods for recovering increased amounts of petroleum from the underground reservoirs.

During the late part of 1917 and throughout 1918, the activities of the Bureau of Mines on petroleum and natural gas were devoted in large measure to problems of the war. The personnel of the petroleum division, because of their training and their knowledge of oil and gas matters, were in a position to give immediate service to the newly created United States Fuel Administration and to the armed forces of the country. Throughout the war-time period, in addition to the performance of a variety of confidential technical work, a determined effort was made to emphasize the prevention of waste of oil and gas, and their more efficient utilization.[10]

At the end of the war "the Bureau was confronted with the need of enabling the mineral industries of the country to meet most effectively the inevitable difficulties that would confront them during the transition from the strain of war to the resumption of the normal activities of peace. Investigations in progress were stopped or given a new direction; plans for the future were changed; new investigations were proposed; work was concentrated on those investigations that promised to be most beneficial to the mineral industries during the period of transition and readjustment."[11]

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9 Lewis, J. O., Methods for Increasing the Recovery from Oil Sands: Bull. 148, Bureau of Mines, October 1917, 128 pp.

10 A summarized report of this work is given in Bureau of Mines Bull. 178-C, War Work of the Bureau of Mines: Petroleum Investigations and Production of Helium: June 1919, pp. 63-88 (out of print).

11 Ninth Annual Report of the Director of the Bureau of Mines for the fiscal year ended June 30, 1919, p. 9.

As examples of peace-time activities, immediately following the war, the petroleum division of the Bureau evolved a new method for estimating the future and ultimate production of oil properties,[12] under the then existing practices of so-called "normal decline," and a comprehensive study of underground conditions in oil fields resulted in a publication[13] on that subject which is a continuing standard reference for all production engineers.

During this time also, as examples of the chemical studies, a widely recognized authoritative report was made on the quality of gasoline marketed in the United States,[14] and a system of analyzing crude oils was developed which is the basis for the Bureau's reports on its surveys of crude oils produced throughout the world.[15]

With the passage of the so-called "Leasing Act" of February 25, 1920, Bureau engineers prepared operating regulations for oil and gas leases under this act to protect Government lands against waste and damage in the development of oil and gas, and from March 3, 1921, to July 1, 1925 (on which date the Bureau of Mines was transferred from the Department of the Interior to the Department of Commerce), all supervisory and regulatory work pertaining to the drilling for and the production of oil and gas on public lands was vested in the Bureau of Mines.

Growth of the Industry

The petroleum industry has developed its capital investment from the one-half billion dollars, reported for 1914, to a total in 1932 reported to be 12 billion dollars. Concurrently, the natural-gas industry has grown from the status of an almost unwanted appendage of the oil industry to a self-supporting business of national importance. In 1930, the natural-gas industry was conservatively estimated to have expended $500,000,000 for expansion alone; and in 1931, through its transmission lines (some extending 1000 miles or more across the country) it delivered gas valued at $392,816,000 at points of consumption. Although there was an appreciable indicated decline in consumption for 1932, the estimated value for that year was well over $350,000,000.

During this epoch, the problems of oil and gas have changed. As some problems have been solved, many others have appeared and the whole economic and technical structure of the oil and gas industries and the public interest therein have grown more complex.

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12 Beal, Carl H., The Decline and Ultimate Production of Oil Wells, with notes on the Evaluation of Oil Properties: Bull. 177, Bureau of Mines, 1919, 215 pp. Later followed by Cutler, Willard W., Jr., Estimation of Underground Oil Reserves by Oil-Well Production Curves: Bull. 228, Bureau of Mines, 1924, 114 pp.

13 Ambrose, A. W., Underground Conditions in Oil Fields: Bull. 195, Bureau of Mines, 1921, 238 pp.

14 Hill, H. H., and Dean, E. W., Quality of Gasoline Marketed in the United States: Bull. 191, Bureau of Mines, 1920, 275 pp.

15 Dean, E. W., Hill, H. H., Smith, N. A. C., and Jacobs, W. A., The Analytical Distillation of Petroleum and Its Products: Bull. 207, Bureau of Mines, 1922, 82 pp.

Changing Scope of Activities

Today engineers are confronted with questions regarding the economic as well as the physical conditions that are involved in drilling to and producing from deep-lying formations, which less than 10 years ago were only conjectured; the effect of production control on ultimate recovery; the interpretation of so-called "bottom-hole" or subsurface pressures; the amount of gas dissolved in the oil under reservoir conditions; the flow of oil and gas through the porous rocks of the reservoir and the related problems of well spacing; the many phases of unit operation and the gas-oil-energy attributes of the structures; the widespread production, distribution, and use of natural gas; the manufacture of motor fuels and lubricants by processes which had not been conceived of in 1914 and 1915; and the complications of the technical phases of taxation.

As the industry has changed through growth and expansion, so have the field of activities and some of the types of study of the Bureau of Mines changed. From the very natural result of the general dissemination of basic knowledge, the Bureau no longer is required actually to tutor by personal instruction in the technique of oil and gas production in the same manner as in former years. Technical institutions and universities have developed courses in petroleum technology with the result that an increasing number of graduate engineers, who have specialized in oil and gas studies, have been available for employment by the oil and gas companies. Also, many former Bureau of Mines engineers have accepted responsible positions in the petroleum and natural-gas industries and have done much to implant the principles of good engineering and scientific thought as applied to petroleum and natural gas in the organizations which they now serve. Many companies have their own research organizations, both in the laboratory and in the field, and some types of laboratory research are conducted at universities and other technical institutions.

Recognizing this growth and change, the Bureau of Mines for several years has concentrated its efforts on studies of a fundamental nature that apply throughout the whole industry and that can not reasonably be made by individual companies or others. In so doing it is closely following the policy established 18 years ago to do pioneer work on problems that have broad, if not national or international, significance. Some of the early problems were so fundamental that the general plan of attack has not radically changed. These researches of the continuous type should be perpetuated in order to accumulate the knowledge that is required for proper functioning of the industry.

This period of change in some major activities of the Bureau was perhaps first definitely evidenced when in 1926 the oil-recovery laboratory at the Petroleum Experiment Station, Bartlesville, Okla., was organized with a working program and outline of procedure as definite as any basic research project will permit, and the former practice of having one or more expert drillers in the field was discontinued shortly thereafter.

A research project which requires the development of a philosophy is never undertaken by the Bureau until a careful canvass of its need has been made. For example, several years of planning preceded the establishment of the oil-recovery laboratory as the nucleus of a definite project. Again, before work was started to determine the amount and character of gas dissolved in oil in natural reservoirs, personal discussion or correspondence was held with representative engineers in all parts of the country in order to obtain their views regarding the need for such work. Conversely, when a problem reaches a state of completion where sufficient data have been compiled to enable the industry to make practical application of the results, a report of the findings is written for general distribution, and active work on the problem is suspended. This procedure was typically true of the problem of cementing wells, when some years ago it became evident that the industry had accepted the results of findings of the Bureau on that subject and was using them effectively to protect against underground waste of oil and gas and was amplifying the technique as occasion demanded, due to peculiar local conditions or new developments. However, current information on suspended problems is gathered and kept on file for use as needed in the general dissemination of information or for revision of earlier published work.

Facilities for Study

At the present time (May 1933) the petroleum and natural-gas division of the Bureau of Mines has a staff of approximately 50 technical men and a small force of clerks, mechanics, and skilled laborers. Headquarters for fieldwork are maintained at the Petroleum Experiment Station, Bartlesville, Okla., and at the petroleum field offices at San Francisco, Calif., Dallas, Tex., and Laramie, Wyo.

The Petroleum Experiment Station was established in 1917 upon a site containing 5 acres of land given by G. B. Keeler of Bartlesville. Two brick buildings, one an administration and office building and the other a chemical laboratory, were erected through the efforts of the Chamber of Commerce of Bartlesville, Okla., in providing $50,000 for that purpose. During the life of the station, the State of Oklahoma has generously contributed cooperative funds to aid in its maintenance and operation and for the conduct of oil and gas investigations which pertain more specifically to the industry of that State. Various other studies on relatively "short-time" problems have been conducted at the Petroleum Experiment Station and at the petroleum field offices under the Bureau's organic provision which enables it to enter into formal cooperative agreements with agencies that are interested in technologic advancement in the mineral industries.

The San Francisco office has been maintained since the Bureau of Mines was organized in 1910. At first it was primarily a field laboratory where fuel oils, purchased by the Federal Government, were tested. Later, as the interest in petroleum technology grew more pronounced, the San Francisco office became the western center of the Bureau of Mines on all matters pertaining to oil and gas. Phases of the Bureau's petroleum research projects under study at San Francisco will be discussed later in the paper.

The Dallas (Tex.) office was opened early in January 1920. The Dallas Chamber of Commerce was instrumental in bringing about the final selection of that city as the headquarters for petroleum engineers of the Bureau of Mines making studies in the oil fields of Texas, Louisiana, and Arkansas. The engineering field studies, made by the Dallas office of typical fields in all parts of the Gulf-Southwest, are related definitely to the production - research projects being conducted at the Bartlesville Station and at the other field offices.

The Bureau's work at the Laramie, Wyo., office began July 1, 1924, at the request of the Rocky Mountain operators and the State of Wyoming. The initial work there was devoted almost exclusively to production problems. With the completion and publication of the results of a study dealing with paraffin and congealing oil, interspersed with many shorter reports, the Bureau suspended active work on production problems at that office, and in January 1930 the personnel began a chemical and refining study to evaluate the future economic importance and refining values of the black oils of the Rocky Mountains.

This change in type of work was decided upon only after a thorough personal canvass of the changed conditions in the Rocky Mountain area clearly indicated to the Bureau that production problems were becoming progressively less important, and that the black-oil resources were a growing factor in the economic welfare of the region. The relation of this work to other refinery problems of the Bureau will be discussed further on in the paper.

Interrelationship of Research Problems

In the earlier work of the division, investigations were grouped under the three general headings of petroleum technology, chemical technology, and engineering technology, but the individual engineers and scientists worked more or less independently of each other. Correlation was maintained through the office of the chief petroleum technologist, and every man assisted his associates by constructive criticism of manuscripts and reports. This system proved to be successful and has led to the frequent statement that Bureau of Mines reports are among the most critically reviewed pieces of writing that appear in the technical literature. Because of the wide scope of his own problem and the many subjects theretofore untreated, it was logical that each man should follow his own course in collecting data and putting the results of his findings into published form where they could be referred to and used. The result is reflected by the nearly 500 publications on oil and gas that have been issued by the Bureau of Mines. This number does not include the many papers given before technical and other societies and specially prepared articles printed in the technical press.

The interrelationship of the several problems, however, both in refining and production, has been sensed for several years. This condition was evidenced first in reference to the chemical work, and after the petroleum laboratory was transferred from Pittsburgh, Pa., to Bartlesville, Okla., in 1923, the chief of the chemical section at the Bartlesville station served as supervising chemist for the whole division, devoting much of his effort to

coordinating the problems then under study. In 1930, changed conditions showed the need for even closer correlation, and definite steps were taken to establish a direct and unified supervision of all work of the division dealing with petroleum chemistry and chemical engineering as pertaining to the refining of oils.

As in the refinery work, a common fundamental relationship was proved to exist and to govern the problems relating to the production of oil and gas. During the last 3 years especially, the several production-research projects have been conducted in accordance with a definite plan whereby the engineers in charge of the several studies meet in frequent conference and discuss their interrelated and mutual problems.

This interrelation of research problems will be developed further as the several problems are discussed.

Problems

For convenient treatment, the technical problems of the Bureau of Mines, pertaining to oil and gas, are discussed in the following pages under four general groups: (1) Petroleum chemistry and refining; (2) production of gas and oil, including related problems of pipe-line transportation of natural gas; (3) engineering field studies, and (4) special engineering problems.

(1) PETROLEUM CHEMISTRY AND REFINING

The major problem of the refinery section deals with sulphur compounds and other harmful substances in crude oils and their segregation and removal by better methods of fractionation and subsequent chemical treatment. To assure that the work in one laboratory would not be unnecessary duplication of work performed in the other laboratories but that each would supplement and augment the others, the supervising engineer at Laramie, Wyo., who is in direct charge of the studies of high-sulphur oils of the Rocky Mountains, is also held responsible for the progress of the work on refining the naphthene - base oils of California in the laboratory at San Francisco, Calif., and for the fractionation tests in visible distillation equipment and the other problems pertaining to sulphur in oils at the Bartlesville, Okla., station. Progress under this arrangement, since it was instituted in 1930, has proved the feasibility of the plan.

"Black Oils" of the Rocky Mountains

The study of the "black-oil" resources of Wyoming and the thermal-decomposition of Rocky Mountain high-sulphur crude oils is a phase of the major refinery project of the Bureau dealing with sulphur in petroleum.

The economic importance of this study to the whole Rocky Mountain area is clearly indicated by the fact that the production of light, low-sulphur crude oils in Wyoming has been decreasing since 1923. In 1930, the light-oil production about equalled the potential production of black oils, which a recent survey shows is between 36,000 and 46,000 barrels per day for wells already drilled. Actually, the black-oil production in 1932 was approximately 5,000 barrels per day. Unless new light-oil fields are discovered, more and more black oil must be refined to supply the regional demand for petroleum products.

The increasing importance of the black oils of the Rocky Mountains, as a source of refined products with which to meet a somewhat localized market, is evidenced by the activity of the Wyoming and Montana refiners in expanding their black-oil refining capacities during the latter part of 1932 and the first half of 1933.

The published conclusions so far developed in the study of high-sulphur crude oils of Wyoming are given in Bureau of Mines Technical Paper 538[16] (released October 25, 1932) as follows:

> The results of this survey indicate that most of the "gasoline and naphtha" fractions from Wyoming black oils, which constitute the portion of the crude used in the manufacture of motor

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16 Thorne, H. M., and Murphy, Walter, A survey of the High-Sulphur Crudo Oils (Black Oils) Produced in Wyoming: Tech. Paper 538, Bureau of Mines, 1932, 56 pp.

fuel, are deficient in light ends and have a sulphur content greater than 0.1 percent. These fractions will need blending with light material in order to meet distillation requirements and will require desulphurization treatment in order to meet the sulphur specifications for motor fuel.

These crudes can be made to yield more gasoline by cracking than is shown by the analyses in this report, but due to their high-sulphur and carbon-residue content, cracking offers difficulties of operation which make it necessary to design especially adapted equipment.

As this survey indicates, the black oils contain greater amounts of lubricating-oil stocks than many of the well-known lubricating-oil crudes, but the lubricating-oil stocks from these black oils will require extensive and expensive treatment due to their high-sulphur and asphaltic material content.

A series of tests on cracking typical black oils from the Oregon Basin, Garland, Grass Creek, and Dallas-Derby fields, and treatment of the products, is now being conducted at the Laramie (Wyo.) office. The tests are being made in a high-pressure experimental still having a capacity of 5 gallons, tested to 2,000 pounds per square inch, and operated at pressures up to 500 pounds per square inch. The object of the work is to determine the effect of cracking on the sulphur compounds and to determine how this cracking influences the distribution of sulphur in the various fractions.

It has been found that different pressures and temperatures have an effect on the decomposition of the sulphur compounds in the oil and on the yield of distillates, and some indications of optimum conditions of producing motor fuels from these crude oils by cracking are being obtained. The preliminary treating experiments on the distillate indicate the advantage of treating at low temperatures, but so far the amount of acid required is above the economic limit. Through the cooperation of a Rocky Mountain refiner, some detonation tests have been made on motor fuels obtained in the experimental work, which indicate that straight-run gasolines from black oils have a rather low octane rating.

Fractional Distillation of California Lubricating Crude Oils

In 1929, a study of the removal of asphalt and sulphur compounds from California lubricating crude oils by fractional distillation was started in the San Francisco laboratory.

This problem was undertaken at the completion of the engine service tests of internal-combustion engine lubricating oils made from California crude petroleum.[17]

17 The engine-service tests were initiated in 1923, in cooperation with the American Petroleum Institute, and reported by Gavin and Wade in 1926 in Bureau of Mines Technical Paper 387; followed by a companion study of the relationship between volatility and consumption of lubricating oils in internal-combustion engines, reported by Wade and Foster in Technical Paper 500 (published in 1931).

The growing use of vacuum distillation and efficient fractionation pointed to those refining methods as being superior to the prevailing refinery practices in California of partly removing these deleterious substances by chemical treatment and filtration.

To that end a laboratory batch still and a visible-action fractionating column, to be operated under vacuum, were designed and built to produce lubricating distillates under as nearly ideal conditions as possible in order that a subsequent study of the characteristics of the distillates could be made. The fractionating column is so arranged that it can be operated under adiabatic conditions, and localized or premature cracking of the oils in the still has been reduced to the minimum by a method of securing even heat transfer to the oil.

Three typical oils (Coalinga, Midway, and Kern) yielding California lubricating distillates were distilled at an absolute pressure of 1 mm. of mercury at the top of the column and 10.5 to 13 mm. in the still. A description of the still and column, the method of operation under vacuum, and tabular data on the physical and chemical characteristics of the close-cut fractions of the tests are given in Report of Investigations 3159.[18]

Since the initial report was issued, the batch still has been replaced by a pipe still in order that continuous operation can be maintained.

To evaluate the several products obtained in the still and fractionating tower, it was necessary to develop a method and apparatus to measure the stability of the lubricating distillates. Refiners on the Pacific coast have shown much interest in this method, developed by necessity out of the vacuum-distillation study, whereby it has been possible to assign numerical values to lubricating oils which will represent their ability to resist deterioration due to oxidation, polymerization, and condensation. The stability-test method is now being thoroughly reviewed and checked, both within and outside the Bureau, not only on western oils but on eastern oils as well.

Chemistry and Refinery Studies at Bartlesville, Okla.

At the Petroleum Experiment Station, Bartlesville, Okla., the major "sulphur-in-petroleum" problem has been under study, without interruption, since the petroleum laboratory was moved from Pittsburgh, Pa., in 1923. In fact, the sulphur problem has been of major importance to the petroleum industry since 1880-1885, and studies of many phases of the problem have been made by Bureau chemists and engineers.

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18 Guthrie, Boyd, and Higgins, Ralph. Laboratory Batch Still and Fractionating Column for Production and Study of Lubricating Distillates under Vacuum: Rept. of Investigations 3159, Bureau of Mines, 1932, 18 pp.

A phase of the sulphur problem has been studied in recent years by the use of a visible-action continuous-distillation apparatus for laboratory study of fractionation. This device was described in Report of Investigations 2892, published in 1928, and was an exhibit at the International Petroleum Exposition at Tulsa, Okla., in 1929, where it attracted the favorable attention of many refinery technologists from this and foreign countries. The apparatus will form the nucleus of the Bureau's exhibit of petroleum refining by fractional-distillation methods at the "Century of Progress" in Chicago.

In a more recent study of the characteristics of Winkler County (Tex.) crude oil containing 1.49 percent sulphur, it was found that by proper fractionation and by dividing the gasoline into two or more fractions and treating them according to their several requirements, gasoline satisfactory as a motor fuel could be produced from this crude oil without as much chemical treatment and resultant losses as if it were produced from the crude oil in one stream. Complete results of three series of runs on this oil in the visible-action laboratory apparatus are given in Technical Paper 505.[19]

The Bureau has made a long-time study of various methods of refining light petroleum distillates. For some years its refinery engineers have maintained that proper fractionation of the gasoline yield of a pressure distillate will, with most distillates, decrease the amount of chemical treatment that will be required by the gasoline to meet trade requirements. The writers of Bulletin 333,[20] which describes various refining and treating methods, conclude that although "each crude and each pressure distillate has its own characteristics, and it is impossible to formulate a set of rules as to how much of each crude or each distillate should be taken overhead in the vapor phase and how much should be taken off the tower in the side stream," the beneficial results of the proper use of fractionation can not be denied. They recommend also that refiners investigate the possibilities of lessening the need of treating their products chemically by greater use of efficient fractionation and thereby reduce the attendant losses of valuable petroleum products.

Basing procedure upon the previously determined results of fractionating light petroleum distillates, and in line with other sulphur studies of the Bureau, work was started on high-sulphur paraffin-base lubricating oils at the Bartlesville station in the belief that the amount of sulphur compounds which cause bad color and odor in the oils can be reduced in the process of manufacture by means of proper fractionating equipment. To conduct this study, a bubble tower operating under vacuum has been designed and built. The tower, 6 inches inside diameter and 9 feet, 8 inches high, is so constructed that action within it may be observed through inspection ports. The tower assembly, together with a pipe still of the electric muffle-furnace type, condensers,

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19 Espach, Ralph H., and Rue, H. P., Influence of Fractionation on Distribution of Sulphur in Gasoline: Tech. Paper 505, Bureau of Mines, 1931, 24 pp.

20 Rue, H. P., and Espach, Ralph H., Refining of Light Petroleum Distillates: Bull. 333, Bureau of Mines, 1930, 111 pp.

receivers, vacuum pumps, a refrigeration unit, and necessary accessories, all of which are supported on a steel framework, is over 9 feet high, 8 feet long, and 3 feet wide. Preliminary runs have been made and a vacuum as low as 3 mm. absolute pressure can be maintained on the whole system. It is hoped that conditions will permit a series of runs to be started in the near future, in which various refining problems encountered in the manufacture of paraffin-base lubricating oils will be studied.

Another phase of the sulphur problem has been a study at Bartlesville of methods for determining the sulphur in gasoline. In Technical Paper 513,[21] the authors, who studied a method of sulphur determination, concluded that "in general, the determinations of sulphur (made by them) proved quite clearly that the Edgar and Calingaert method, with the use of a sensitive indicator and provision against undue evaporation losses and improper lamp operation, is a very satisfactory method of obtaining check results and, in the case of some compounds, of accurately determining the percentages of sulphur in gasoline."

Wax-Distillate Problems

One of the administrative problems confronting the Bureau of Mines is to make available for use by the industry material that has taken years to collect. One such case before the Bureau is a study of methods of handling wax distillates in order to minimize the bad effects of amorphous wax. The problems involved in their wax content hinder large quantities of crude oil from becoming more valuable for the manufacture of high-grade lubricating oil. Work on a manuscript dealing with the manufacture and refining of wax was started by Wyant and Marsh in 1925. These authors prepared some excellent material before they resigned from the Bureau of Mines several years ago, but when an opportunity presented itself to publish the results, their work needed to be checked against recent developments in the art.

Prior to July 1, 1932, an engineer of the Bureau visited refineries to investigate present methods of handling wax, but his work has been delayed. Because the refinery group at Bartlesville is limited to three men, which does not permit conducting more than one active problem at a time, the definite decision has been made that pending work on fractionation of lubricating distillates in the large-scale vacuum-distillation unit, which has been mentioned, will not be carried forward until the revision of the paper on wax and wax distillates is completed. The industry will thereby be given the benefit of this investigation, extending over a period of years, and at the same time reducing the number of refinery problems under study - now too large for the force of technical men that can be assigned to them.

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21 Espach, Ralph H., and Blade, O. C., Studies on Determination of Sulphur in Gasoline: Tech. Paper 513, Bureau of Mines, 1931, 22 pp.

Synthesis of Motor Fuels from Natural Gas

Not only has the art of manufacturing commercial products from petroleum experienced a revolution of ideas during the history of the Bureau of Mines, but ways of manufacturing products from natural gas, other than by the now conventional methods of natural-gasoline extraction, are no longer considered visionary.

Recognizing that large quantities of natural gas are being wasted because no ready market is available, a study was begun at the Bartlesville station in 1928 to develop possible methods of utilization through the synthesis of motor fuels from natural gas, using the principle of thermal-decomposition of natural gas, with temperature, pressure, and flow rate as the major variables. The aromatic hydrocarbons, especially benzol, are well-known antiknock motor fuels, and their economic manufacture from natural gas, especially from gas composed chiefly of methane, would provide a means of using much gas that is now wasted.

Various investigators had found points in common regarding the synthesis of light oils from methane, but they were not in agreement on many points, especially in relation to the temperatures necessary for maximum yield. Consequently, the Bureau, after a thorough review of the available literature, conducted work to devise apparatus and develop technique for the pyrolysis of methane. The preliminary report on this subject was given at Indianapolis before the American Chemical Society on April 3, 1931, and in October 1931, Report of Investigations 3143,[22] the first of a series of papers on the production of motor fuels from natural gas, was published by the Bureau.

Since the initial report was published, the laboratory experiments have been extended to include the processing of natural gas in a large-scale laboratory unit where calorized KA2 steel tubes, 6 feet long, with diameters of 1, 1½, and 2 inches have been used. Yields of 3/10 gallon per thousand cubic feet of gas have been obtained, and certain optimum conditions of operation have been established. In order to obtain a maximum conversion to light oils, a study also is being made of catalytic methods of hydrogenating the tars, which are produced in the pyrolitic treatment, by utilizing for this purpose the hydrogen present in the residue gas from the process. To complete this problem it would be necessary to build a test-plant unit for continuous and fairly large throughput.

Removal of Hydrogen Sulphide from Natural Gas

The hydrogen-sulphide problems of the petroleum and natural-gas industries have a three-phase relationship: namely, safety, corrosion, and removal by treatment. The first two phases will be discussed later in this paper. Methods of removal from natural gas more nearly fall within the group of chemistry and refining problems than do safety and corrosion, although actually one phase of the subject can not be adequately discussed without reference to the others.

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22 Smith, H. M., Grandone, Peter, and Rall, H. T., The Production of Motor Fuels from Natural Gas - 1. Preliminary Report on the Pyrolysis of Methane: Rept. of Investigations 3143, Bureau of Mines, 1931, 12 pp.

A number of commercial processes have been developed for removing hydrogen sulphide from natural gas, but the problem is by no means solved in all its ramifications. The results of a special study made by the Bureau of Mines show that hydrogen sulphide can be removed economically from "wet" natural gas by means of a lime and salt solution, by contacting the gas and the solution in the proper scrubbing device. Both lime and salt are relatively cheap chemical reagents, and at one commercial plant where this method was tried, satisfactory economies in operation were effected.

In Report of Investigations 3178,[23] wherein this method is described, the further suggestion is made of the possibility of converting certain natural oil-field waters to the proper treating solution by the addition of relatively small amounts of certain chemicals, thus reducing the cost of the chemicals necessary to remove hydrogen sulphide from natural gas.

The Chemistry of Oils

Not all work on oils is directed to refining methods. Following the work of Allen and Jacobs,[24] the crude-oil surveys to determine the physical and chemical properties of petroleums and their relative values as sources of commercial products were extended, and today the Bureau of Mines collection of crude-oil samples is one of the largest. The Bureau's method of analysis,[25] developed as post-war work to meet the industry's requirements of that transitory and readjustment period, makes possible the comparison of different crude petroleums on a reproducible basis. The Bureau's classification,[26] in accordance with the "base" of an oil, is known and recognized wherever analyses of crude oils are considered.

More recently, because of the generally recognized need for information on the subject, much attention is being given to possible relationships between geologic depth and chemical characteristics of the oils. Such a study is being made of oils produced from sands of the eastern United States, and a manuscript discussing oils from the Eastern Hemisphere with reference to producing formations is now being reviewed by competent authorities in those fields. Many of the samples to be reported in the bulletin on crude oils of the Eastern Hemisphere were difficult to obtain, and only through the willingness of foreign countries and other agencies to cooperate with the United States Government were the samples delivered for analysis and subsequent report. The

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23 Rue, H. P., The Use of Lime in a Salt Solution for Removing Hydrogen Sulphide from Natural Gas: Rept. of Investigations 3178, Bureau of Mines, 1932, 8 pp.

24 Allen, I. C., and Jacobs, W. A., Physical and Chemical Properties of the Petroleums of the San Joaquin Valley, California: Bull. 19, Bureau of Mines, 1911, 60 pp.

25 Dean, E. W., Hill, H. H., Smith, N. A. C., and Jacobs, W. A. The analytical distillation of petroleum and its products: Bull. 207, Bureau of Mines, 1922, 82 pp.

26 Smith, N. A. C., The Interpretation of Crude Oil Analyses: Rept. of Investigations 2806, Bureau of Mines, 1927, 20 pp.

work of collecting samples of foreign crude oils has been in progress for more than 10 years. In connection with other studies, representative samples have been collected in the Oklahoma City field, and the analyses have been reported.[27] One report has been issued giving analyses of crude oils produced in East Texas,[28] and a subsequent report, in preparation, will discuss the characteristics of these and other East Texas oils in relation to their probable source beds.

A major research related to the crude-oil surveys pertains to the chemistry of crude petroleum, as shown by the Bureau of Mines Hempel method of analysis. Although data are being accumulated on this subject through the analyses of representative oils, no opportunity has been given to prepare this material into a bulletin for publication.

This problem is typical of many that are under study by the Bureau of Mines. It was begun in the early years of research on petroleum, and like many others it is so fundamental that the general plan of attack has not radically changed. Many of these researches of the continuous type should be perpetuated in order to accumulate the knowledge that is required for the proper functioning of the industry.

Gasoline Surveys

Other studies pertain to marketed products. The surveys of motor gasolines by the Bureau of Mines date back to work in 1911 on comparisons of gasoline and alcohol as fuels for internal-combustion engines. It soon became evident that there was a dearth of published information on the quality of different motor fuels sold throughout the country. Following an initial report[29] on the physical and chemical properties of gasoline marketed in 1915, more extensive surveys were made in April 1917 and April 1919. Subsequently, the semiannual motor-gasoline survey of the Bureau was developed in response to an ever-growing demand for this type of technical information on the part of the manufacturers of motor fuels.

With the issuance of the twenty-first semi-annual motor-gasoline survey, it became evident that these surveys either must be discontinued or made more extensive in order to meet any objection that they were not showing a true cross section of the characteristics of marketed motor fuels. It was finally decided to extend the surveys, and the number of cities where samples were obtained was increased, and the number of samples collected was doubled to approximately 300. Also, samples of the gasolines tested in the routine

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27 Garton, E. L., Analyses of Crude Oils from the Oklahoma City Field, Okla.: Rept. of Investigations 3180, Bureau of Mines, 1932, 29 pp.

28 Garton, E. L., Properties of Typical Crude Oils from the East Texas field: Rept. of Investigations 3130, Bureau of Mines, 1931, 7 pp.

29 Rittman, W. F., Jacobs, W. A., and Dean, E. W., Physical and Chemical Properties of Gasoline Sold Throughout the United States During the Calendar Year 1915: Tech. Paper 163, Bureau of Mines, 1916, 44 pp.

laboratory of the Petroleum Experiment Station were sent to the Bureau of Standards, where through the cooperation of that Federal agency, detonation tests were made and the results reported, giving the first published survey of octane numbers of gasolines.[30]

The making of the gasoline surveys throughout the country entails a large amount of expense and work, particularly where a Bureau representative actually collects the gallon samples at service-station pumps in 20 cities, and increasing need of retrenchment forced the Bureau of Mines to discontinue the surveys with the issuance of reports on the survey of August, 1931.[31]

Since information on the properties of commercial motor fuels throughout the United States was desired by the industry, a canvass was made of representative oil companies who make gasoline surveys for their own information in their own marketing districts. It was thought that if these companies could supply the necessary information, the Bureau could analyze these results in much the same way as they had analyzed the results of their own analyses and continue to publish reports thereon, retaining, of course, the previous practice of keeping the identity of samples unrevealed. The response from the oil companies was indicative of their continuing interest. In fact, a dozen or more companies submitted data sheets, but these had to be returned to the senders, because in the first part of July 1932 it became evident that the Bureau would not be able to publish this information, for the reason that the Federal appropriation for oil and gas work had been greatly reduced. The only possible way by which these reports may be continued at the present time, and the intervening gap filled, is a definite action on the part of the industry through some of its agencies to arrange for the publication of these data which the Bureau is anxious to perpetuate in permanent record.

Routine Laboratory

As a very important adjunct not only to the refinery and chemical section but to the oil-field investigations and research problems as well, a routine laboratory has been maintained at the Petroleum Experiment Station, Bartlesville, Okla., since 1923, when the Bureau's work on the chemistry of oils was transferred from the Pittsburgh (Pa.) Experiment Station, in order that the technologic studies of all phases of petroleum by the Bureau of Mines would be more centralized. In this laboratory, analyses of samples of crude oils, distillates, motor fuels, oil-field waters, and natural gas are made.

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30 Kraemer, A. J., and Lane, E. C., Twenty-second Semi-Annual Motor Gasoline Survey; Additional Data: Rept. of Investigations 3092, Bureau of Mines, 1931, 32 pp., including a section on detonation characteristics of motor fuels by the United States Bureau of Standards.

31 Kraemer, A. J., and Lane, E. C., Motor-Gasoline Survey, August, 1931. Part 1 - Specification Data: Rept. of Investigations 3162, Bureau of Mines, 1932, 24 pp.
Kraemer, A. J., Lane, E. C., and Garton, E. L., Motor gasoline survey, August, 1931. Part II - Additional Data, with a section by N. R. White and H. K. Cummings on Detonation Characteristics of Motor Fuels: Rept. of Investigations 3175, Bureau of Mines, 1932, 31 pp.

Frequently, samples are submitted by individuals requesting an analysis of the material for commercial reasons. The Bureau's petroleum laboratory is not a commercial laboratory, although upon request it does a small amount of routine testing for other Federal agencies. The general practice is for the Bureau to collect or request collection of its own samples, and the senders of unrequested samples are referred to recognized commercial laboratories who are qualified to do the work in question.

Summary Statement

The foregoing review of the status of some of the Bureau's present studies, as they pertain to the chemistry and refining of petroleum and its products, is by no means complete. The reader should have gained, however, from text and footnote reference, a general idea of the types of problems under study and a suggestion of the results. Selected lists of Bureau of Mines publications pertaining to oil and gas are available to those who desire more complete reference to the published work on the several investigations.

(2) PRODUCTION OF GAS AND OIL

(Including related problems of pipe-line transportation of natural gas)

The same physical laws of flow, and certain fundamental relationships based upon these laws, hold for all fluids and fluid mixtures traveling in horizontal pipe lines, in vertical pipes, and in the intricate passages made up of the connecting spaces in porous media. Naturally, the character of the fluid and the walls of the conducting spaces change the form of the mathematical expressions describing the flow under a set of specified conditions. For this reason, engineers throughout the petroleum and natural-gas industries are endeavoring by many experimental methods to determine suitable numerical values for certain constants and exponents that may be applied in making computations. The objective is to obtain reasonable results without unnecessary refinements which may become meaningless in the light of other controlling factors.

From the foregoing statement, it is seen that studies of pipe-line transportation of natural gas have much in common with studies of flow relationships in the production of gas and oil.

The Annual Report of the Director of the Bureau of Mines for the fiscal year ended June 30, 1931, stated:

> The present problems pertaining to (1) flow of gas through pipe lines, (2) estimating reserves, (3) gaging and controlling natural-gas wells, (4) increasing the recovery of oil and related problems of flow of oil, gas, and oil-gas mixtures through porous media, (5) solubility of natural gas in crude oils, (6) vertical flow in wells, of which knowledge of flowing temperatures and pressures at various positions in the wells is so essentially a part, (7) injection of gas into partly depleted sands, and (8) migration of gas in reservoir sands are all essentially one large group of correlated research within the bureau.

For administrative purposes, these production problems are placed in four working sections - natural gas, oil recovery, vertical flow, and gas solubility - each section of which is connected actually, as well as diagramatically in the organization chart, to each of the other sections. As data are developed on any one of these problems, they strengthen the structure of the whole project of production research. For example, in the Bureau's study of pipe-line transportation of natural gas, flow relationships of the upstream and downstream pressures were found to have direct application not only to the study of vertical flow of oil-gas mixtures in wells but also to the study of oil-recovery problems which involve migration of oil and gas through sands and the spacing of wells.

Natural Gas Problems

In the opinion of the Bureau of Mines, research problems of the natural gas industry and those of the petroleum industry are not subject to definite separation. This fact was clearly demonstrated in the cooperative report of the Bureau of Mines and the American Petroleum Institute entitled "Function of Natural Gas in the Production of Oil."[32] However, certain studies develop from conditions which are peculiar to the recovery and handling of each of these companion hydrocarbons.

The work on natural gas extends back to the Bureau's organization. For the past 10 years the problems have had a definite place in the industry's program through formal cooperation with the Natural Gas Department of the American Gas Association (formerly the Natural Gas Association). This work dates back to preliminary but very essential and basic tests by Cattell and Rawlins on the flow of gas through small orifices.[33] That work has now been extended to orifices of large diameter (up to $3\frac{1}{2}$ inches) and a bulletin is in preparation which will cover all pressure conditions that would be found in the field or in experimental study.

On the basis of the small orifice relationships, a Nation-wide study of leakage losses from high-pressure natural-gas transmission lines was made, and a generally recognized method of determining these losses was developed which became a part of the code of the American Gas Association. Probably no gas-transmission line of any importance has been placed in operation since 1928, when Bulletin 265[34] was published, that was not thoroughly tested in accordance with the method described in that publication. In fact, on the 24-inch line from the Texas Panhandle to Chicago and on several others, engineers of the Bureau were called upon to give advice in order that the lines would be made safe and loss through leakage would be as small as possible.

Flow of Gas in Pipe Lines

Complementary to the gas-leakage work, another phase of the pipe-line studies was undertaken in 1925 in cooperation with the Natural-Gas Department of the American Gas Association - that of the flow of natural gas through pipe lines, which involves the economical handling and transportation of gas from the well to points of utilization. In their recent report to the pipe-line-flow committee of the association,[35] the Bureau engineers concluded that

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32 Miller, H. C., Function of Natural Gas in the Production of Oil: A report by the U. S. Bureau of Mines and the American Petroleum Institute. Printed by the American Petroleum Institute, 250 Park Avenue, New York, N. Y., 1929, 267 pp.

33 A brief resumé of this work was given before the Southwestern Short Meter Course at Norman, Okla., April 24-26, 1928. (Mimeographed under title: Rawlins, E. L., The Flow of Air and Natural Gas Through Small Orifices Under High Differential Pressure. 15 pp., 5 figs.)

34 Rawlins, E. L., and Wosk, L. D., Leakage From High-Pressure Natural-Gas Transmission Lines: Bull. 265, Bureau of Mines, 1928, 108 pp.

35 The data and findings are published in Rept. of Investigations 3153, Factors Influencing the Flow of Natural Gas Through High-Pressure Transmission Lines, by W. B. Berwald and T. W. Johnson, Bureau of Mines, 1931.

Weymouth's pipe-line-flow formula, when applied to pipe lines free from condensates and operating under steady flow conditions favorable to accurate measurements, will give a volume within a few percent of the measured delivery. From these findings, based upon tests on 29 commercial pipe lines and supplemental tests, it appears that a sufficiently accurate yet relatively simple criterion for the design, construction, and operation of natural-gas pipe lines has been established. The most pertinent conclusions made from this study are being incorporated into the code book of the Natural-Gas Department of the American Gas Association.

For greater accuracy, some investigators hold that the relationships between the coefficient of friction and the variables of the Reynolds' criterion should be applied in the design. However, for practical purposes in designing pipe lines for transmission of natural gas, the Bureau has found that -

> * * * the value of applying many small corrections to pipe-line flow formulas, such as small differences in elevation between the inlet and outlet of the line, reduced diameter, fittings, the effect of bends, right-angle turns, metering stations, drips, type of joints, and the length of pipe between joints is often greatly diminished because of conditions affecting individual lines such as unsteady rates of flow, storage or drainage of gas taking place, the presence of condensates, rust, scale, and other foreign materials in the line, and the relative roughness of the interior walls of different lines. The effect of these conditions is often many times greater than the sum of all the measurable corrections.

A bulletin, augmenting Report of Investigations 3153, and giving the details of this study, is in preparation.

Deviation of Natural Gas from Boyle's Law and Viscosity of Natural Gas

Because of relationships involving the density and absolute viscosity of the gas, supplemental studies of these factors were necessary to correlate intelligently the data on the main problem of flow of gas through pipe lines. The results of these studies have been printed as Technical Paper 539,[36] and a companion paper by the same authors entitled "Viscosity of Natural Gas," Technical Paper 555, is in press.

The casual reader may wonder why engineers on practical research spend much time on what seemingly are problems of pure research and therefore should be dealt with in a highly specialized manner. The answer is that the essential facts which frequently are needed in the practical problem do not exist, so that in order to work out practical solutions, the engineer must make fundamental determinations that cannot be found in any handbook or other scientific writing.

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36 Johnson, T. W., and Berwald, W. B., Deviation of Natural Gas from Boyle's Law: Tech. Paper 539, U. S. Bureau of Mines, December 1932, 29 pp.

The principle of deviation of a gas from Boyle's law is fairly well understood and earlier writings have discussed the subject. However, Bureau engineers found that a procedure of experimental testing or actual measurement of the deviation under different conditions of pressure and temperature was essential, because (1) the deviation is a function of the chemical composition of the gas; (2) gases containing higher hydrocarbons deviate more than those having a larger content of methane; and (3) the chemical composition of a natural gas may change with a change in pressure or temperature due to liquefaction of some of the constituents. A large number of natural gases of different chemical composition were included in the study, making it possible to obtain a fair approximation of the compressibility of any natural gas by comparing its chemical composition with the gases reported in Technical Paper 539.

This work not only has direct application to natural-gas pipe-line-flow problems, but the deviation of natural gas from Boyle's law enters into the calculation of the actual volume of the void space in a sand body and the remaining pressures in the reservoir. It is recognized that the factors of water encroachment and possible errors in pressure and production records may introduce some uncertainties in calculation, but the deviation of natural gas from Boyle's law is likely to have a considerable effect on the estimates of future gas reserves, because when the deviation of the gas from Boyle's law is neglected, the amount of calculated reserve is higher than the actual amount of gas remaining in the structure. Since the publication of Technical Paper 529, the Bureau has had many inquiries from the industry regarding the application of the findings to special oil and gas field conditions.

Until recently, little consideration had been given to the subject of viscosity in solving natural-gas problems. Present-day industrial problems of the petroleum and natural-gas industries are dependent for their solution more and more on the application of the laws of fluid movement, and the report on viscosity, which is written in as nontechnical language as possible, should find wide practical use, equally as wide as has been found for the data developed in the study of deviation of natural gas from Boyle's law.

Methods for Gaging and Controlling Natural-Gas Wells

One of the most outstanding results of the cooperative work done by the Bureau of Mines and the Natural-Gas Department of the American Gas Association pertains to methods of gaging gas-well capacities. This method is commonly referred to as the "back-pressure method." Prior to the active cooperative work on this problem, tests were made in the Chickasha gas field in 1927 by Bureau engineers,[37] at the request of the Oklahoma Corporation Commission, which resulted in the establishment of certain relationships making it possible to evaluate the flow of gas from a well through a certain size of

37 Brandenthaler, R. R., Rawlins, E. L., and Johnson, T. W., Standardizing the Open Flow From Natural-Gas Wells: Rept. of Investigations 2885, Bureau of Mines, 1928, 6 pp. (out of print).

pipe to the equivalent flow through larger sizes, thus reducing the amount of gas blown to the air in an open-flow test. It was realized, however, that improvement in technique and greater savings in gas then wasted in the tests were possible.

In 1929 two reports[38] were published outlining the fundamental relationships involved in the flow of gas through gas sands, and between gas flow and well capacities.

As explained in these reports, the basic thought of the fundamental relation between the three factors (1) rock or formation pressure in the sand, (2) back pressure at the sand face of the well, and (3) the rate of flow from the well, is that the sand of any particular gas well can be calibrated for its capacity to deliver gas from any formation pressure into any back pressure. In other words, there is a straight-line relationship between rate of flow and the difference of the squares of the formation and back pressures for delivery of gas from a formation pressure into a series of different back pressures when the data are plotted on logarithmic paper.

A total of 959 tests has been made on 577 gas wells in Louisiana, Ohio, Oklahoma, the Texas Panhandle, South Texas, and the Rocky Mountain area. In 88 percent of the tests it was possible, for all practical purposes, to draw the relationship as a straight line; in 8 percent, curvature was represented by the relationship; while in 4 percent of the tests, the relationship of the respective plotted points of the test series was irregular.

In some of the recent tests in Texas, it was determined conclusively by this method that cavings behind the casing were interfering with the flowing conditions of the wells. Other applications of the method point to the forecasting of drilling requirements, the design of gathering systems, determination of operating pressures, estimation of costs of production, and judging the advisability of storing gas in sands.

Discussion of many of the tests and their significance might be given, but only one typical example is cited to show how the use of the back-pressure method is saving gas in contrast to the open-flow gaging method where the gas is blown to the air.

In the Texas Panhandle area the equivalent open-flow capacity of 221 wells, which were tested by the back-pressure method, was determined to be approximately 5,485,000,000 cubic feet in 24 hours. It would have required an average of at least 30 minutes, and probably 1 hour or more to have obtained settled flow on these wells. Even with the minimum of a 30-minute flow period,

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38 Pierce, H. R., and Rawlins, E. L., Study of a Fundamental Basis for Controlling and Gaging Natural-Gas Wells. Parts I and II: Rept. of Investigations 2929 and 2930, Bureau of Mines, 1929.

an open-flow gage of these 221 wells would have wasted 114,000,000 cubic feet to the air. These figures do not include underground waste that would have been caused by the wide-open blowing of the wells. The hazards of water coning in the reservoir and detrimental effects on the mechanical condition of the wells because of open-flow tests are well known.

At first the method did not appear to many technicians as being feasible, but there has been a change in viewpoint, and since 1929, many of the tests on public lands, lands of the Osage Nation, and other Indian lands have been in conformance with this method. At an open meeting held some years ago in Oklahoma, it was agreed by the Corporation Commission that the back-pressure method might be used in all fields of the State as an alternate method until further notice. In Texas and Louisiana, the method has been looked upon with favor. In Montana no official action has been taken, but State conservation officers are showing interest. Recently the State of Michigan, desirous of having proper enactments which would help conserve its recently found gas resources, requested the assistance of the Bureau in order that when legislation was enacted the back-pressure method of testing well flows might be adequately provided for. Still more recently the State of Mississippi, faced with natural-gas conservation problems in the Jackson gas field, has called on the Bureau for assistance. Kansas also has asked for aid.

Judging from the foregoing results and the interest shown by the various States, the Bureau of Mines feels that the back-pressure method of determining the potential capacities of gas wells has proved its technical possibilities and its economic value.

Oil Recovery

Methods of production that leave large quantities of unrecovered oil underground are economically unsound and incompatible with true conservation. On this premise the oil-recovery section at the Bartlesville station was organized and its laboratory built and equipped in 1926, in order to determine, through experimental research, methods of increasing ultimate production in partly depleted fields and of prolonging the natural flowing life of wells. Since no research project requiring the development of a philosophy is ever undertaken by the Bureau of Mines until a careful canvass of its need has been made, several years of planning preceded the establishment of the oil-recovery laboratory as the nucleus of a definite project.

There are three active problems under study in the section:

(1) The flow of oil through sands.
(2) Methods of increasing the recovery of oil.
(3) The effect on crude petroleum of air used in repressuring oil fields.

Actually, problem 1 deals with fluid-flow through porous media and has a definite relationship to the work of the natural-gas section on methods of gaging and controlling natural-gas wells. This work also has special bearing on problem 2 dealing with increased-recovery methods of oil production.

Fluid-Flow Through Porous Media

The first series of experiments on problem 1 was conducted in order to obtain knowledge of the relationships of such factors as size and character of grain, permeability and porosity, and their effect on fluid flow. To this end, the natural-gas section and the oil-recovery section jointly performed a series of more than 60 experiments at the Bartlesville laboratory and at a special "hook-up" at a high-pressure gas well in the Texas Panhandle, made available for the experimental work by a gas company whose officials understood the objectives of and saw much promise in the results of the research.

In the initial tests, air and natural gas were flowed through various porous materials under different pressure and temperature conditions and at different rates of flow ranging from 0.2 to 700 cubic feet per minute and at pressures up to 435 pounds per square inch, absolute.[39]

In many respects the flow relationships involving the square of the upstream and downstream pressures were similar to those previously determined for the flow of gases through pipe lines. It was found, however, that there was a gradual transition from viscous to turbulent flow in contrast with the more or less abrupt change which takes place in straight pipes. It appears that the flow of gases through porous media conforms closely to the flow of gases in coiled tubes.

The results of the tests made it possible to determine an exponential equation that provided a helpful means of studying the effect on the flow conditions of such factors as grain diameter, shape of grain, and porosity. In addition, considerable progress was made in developing a method by which the relative capacities of sands to permit fluid flow through them could be compared accurately.

Different investigators had used different methods for expressing this sand quality, although each used the term "permeability." This led to some confusion regarding the meaning of permeability. The results of the Bureau investigation showed that a wide range of permeability factors might be obtained for a single sand, depending upon the rate of flow of the fluid through the sand. The reason for this is the change in flow conditions with changes in fluid velocities. It was evident that if the flow conditions in the various sands could be reduced to a common basis for comparison, the relative capacities of the sands to allow fluids to flow through them would have

39 The flow tubes were designed for a maximum working pressure of 1,500 pounds per square inch, with the view of making tests at this higher pressure on some special well. To date, the higher pressure work has not been performed.

some significance. This basis of comparison was found in the determination of the "mean effective pore diameter" which could readily be made from the experimental data. A value for mean effective pore diameter is based upon the flow capacity of the sand under the conditions of viscous flow.

The determination of a mean effective pore diameter for each of the porous materials used in the tests shows a promising relationship between friction factor and Reynolds' Criterion through which the pipe-line flow formulas can be made more generally applicable to the flow of fluids through porous media.

A description of the equipment and a more complete discussion of the results of the tests are given in a paper by Chalmers, Taliaferro, and Rawlins in Transactions of the American Institute of Mining Engineers, Petroleum Development and Technology, 1932, under the title "Flow of Air and Gas Through Porous Media."

The findings and especially the interpretations of the permeability factor or "mean effective pore diameter" of a porous medium, in conjunction with back-pressure data, have application to well conditions in gaging the ability of a gas well to produce gas under different pressures, in calculating gas reserves, and in solving other practical gas-production problems.

All of the data thus far obtained have an important bearing on the problem of oil- and gas-well spacing. The problem of well spacing as a research project is not new,[40] but as yet it is generally agreed that no definite criteria have been established from which proper spacing patterns for various field conditions can be developed. This is an active problem before the industry, as indicated by studies that are being made by a special committee on well spacing of the American Petroleum Institute and by other organizations.

Engineers of the Bureau have been deeply interested in this problem and have regretted the fact that because of reduced appropriations, experimental work pertaining to the well-spacing program, as a part of the study of flow through porous media and other recovery problems, could not be extended this year to include work in a steel reservoir developed by Mills and his co-workers for studies of radial flow and described in Technical Publication No. 144 of the American Institute of Mining and Metallurgical Engineers, 1928.

However, the work on linear flow is continuing, and when the results on gas and air are satisfactorily analyzed, studies of the flow of oils through sands will be made before undertaking the more complex system involving oil

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40 Beal, Carl H., and Lewis, J. O., Some Principles Governing the Production of Oil Wells: Bull. 194, Bureau of Mines, 1921, pp. 18-21, 25-34.
Brewster, F. M., A Discussion of the Factors Affecting Well Spacing: Trans. Am. Inst. Min. Eng., Petrol. Dev. and Tech. 1925, pp. 37-46.

with dissolved gas and oil-gas mixtures. To further the practical application of the work, cores of natural, consolidated sands are being checked against the unconsolidated sands of the previous experiments.

Methods of Increasing the Recovery of Oil

At a time like the present when the possible rate of production from wells already drilled is in excess of what the market will absorb, the industry is especially interested in methods of curtailing production and bringing about orderly development. However, those who plan for the future realize that the fields now having "flush production" will have to be operated in a different manner in later periods of the life of the fields in order to recover the most oil and help pay out the development investments. The Bureau feels that sight should not be lost of the fact that even with the application of the best knowledge regarding natural flow in accordance with what is known of the function of gas in the production of oil,[41] nevertheless an appreciable quantity of the original oil will remain unrecovered in the sand except for the application of stimulative methods. The Bureau of Mines also has held that no matter how great the undiscovered reserves may be, they are subject to exhaustion, and a forward-looking view requires that knowledge be available when needed that will permit the winning of as much oil from the sands as may be possible. The laboratory investigations being made will give knowledge of the fundamental principles and relationships involved in various methods of increasing the recovery of oil from partly depleted fields.

Several publications[42] have described the equipment and methods used in the oil-recovery laboratory at Bartlesville. Recent work shows that when sand is highly saturated with oil, a wide variation in the rates of natural gas and air injection have only minor effect upon the efficiency of oil recovery by means of the "gas drive." However, for sands of low saturation the efficiency of operation, based upon energy-oil ratios, was greatly reduced with increased gas velocities.

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41 Miller, H. C., Function of Natural Gas in the Production of Oil: A report by the Bureau of Mines and the American Petroleum Institute, printed by the American Petroleum Institute, 250 Park Ave., New York, 1929, 267 pp.

42 Mills, R. Van A., Chalmers, Jos., and Desmond, J. S., Oil Recovery Investigations of the Petroleum Experiment Station of the United States Bureau of Mines: Tech. Pub. No. 144, Am. Inst. Min. and Met. Eng., 1928, 36 pp.
Mills, R. Van A., and Heithecker, R. E., Volumetric and A.P.I. Gravity Changes Due to the Solution of Gas in Crude Oil: Rept. of Investigations 2893, Bureau of Mines, 1928, 15 pp.
Chalmers, Joseph, Recent Studies on the Recovery of Oil From Sands: Trans. Am. Inst. Min. and Met. Eng., Petrol. Dev. and Tech. 1930, pp. 322-328.
Chalmers, Joseph, Nelson, I. H., and Taliaferro, D. B., The Recovery of Oil from Sands by the "Gas Drive": Rept. of Investigations 3035, Bureau of Mines, 1930, 12 pp.

The earlier reported work of the Bureau on the "gas drive" showed that under the conditions of test, without back pressure, air had a greater propulsive effect than gas. Later comparative tests, where back pressures were held on the experimental flow tubes, showed that gas had a distinct advantage over air. Although the energy input was greater for gas than for air, the energy-oil ratio was less, because of the greater production of oil obtained by the use of gas.

It appears that in these flow-tube experiments the reduction of surface tension and decreased viscosity of the oil caused by the gas going into solution under back-pressure conditions tended to increase oil recovery, regardless of the fact that air was found to be more efficient than gas as a propulsive medium. As stated in Report of Investigations 3035 (see footnote 42), "The ideal gas for gas injection is therefore one that will dissolve freely in the oil at the injection pressure and will be liberated from solution by the time it reaches the producing well. Since there is seldom a choice of gas for injection purposes the same conditions may be approached by control of operating pressures."

Cognizant of the fact that laboratory results must have a direct "tie-in" with field operations, the Bureau has made studies of the flow of injected air and gas through oil strata. This work pertains equally to methods of increasing the recovery of oil by stimulative methods and to the recovery of oil by controlled natural flow.

Some time ago a study was made of the results of air repressuring in the Williams Pool, Texas, wherein it was shown that "high-pressuring" of the sands was detrimental because of by-passing and that increased recoveries could be obtained through the careful selection of input wells and the regulation of pressure. Results of this study are given in Technical Paper 470.[43]

More recently a study of the migration of injected gas through oil and gas sands of California has been made. Results of this field study showed that: (1) Injected gas migrates to regions where a lower pressure exists regardless of the structural location of such spaces with respect to the point of ingress of gas; (2) after the pressure within the intercommunicating pore spaces and low-pressure regions in the reservoir sands about injection wells have been equalized, the gas mass moves up the structure; and (3) bouyancy of injected gas alone is too small a force to move the gas mass or "bubble" any considerable distance up the structure, and the principle motivating force behind the up-structure migration of the "gas bubble" is the pressure of encroaching edge water.

43 Hill, H. B., Results of Air-Repressuring and Engineering Study of Williams Pool, Putnam-Moran District, Callahan County, Tex.: Tech. Paper 470, Bureau of Mines, 1930, 69 pp.

Gas injected into steeply dipping reservoir sands in California does not spread out fanwise from injection wells as many engineers and operators had previously supposed, and large bodies of oil-saturated sand between injection and producing wells are often by-passed by the injected gas. As in the case of the Williams Pool study, it was found that high injection rates with non-uniform spreading of injected gas make it difficult to control the direction of migration of the gas. The study proved further that the best results in increasing oil-recovery efficiencies are obtained by maintaining back-pressure on the producing wells to prevent gas from breaking through, and by limiting volume of gas injected through individual wells to a small amount per day so as to build up the reservoir pressure gradually. Report of the detailed findings are given in Report of Investigations 3177.[44]

Effect of Crude Petroleum of Air used in Repressuring Oil Fields

Several years ago when the use of methods of stimulating oil production were at a rather high peak, many questions were asked as to whether or not air in repressuring work would change the chemical composition and physical characteristics of crude oil. Other questions of a similar nature were: Are all types of crude oils oxidized by the use of air? Will the use of air seriously affect the viscosity and surface tension of the crude oil so that it will be more difficult to remove from the sand? Will the use of air change the character of the oil sufficiently to decrease its commercial value? Can the oxidation of crude oil be prevented by the use of inhibitors?

Relatively long-time experiments with oils of different chemical characteristics have been in progress in a specially constructed temperature bath at the Bartlesville (Okla.) station in an effort to answer some of these questions. Some of the tests have been run for a period of 182 days. In order to determine the plugging effect that might result from the formation of precipitated gums, resins, and asphaltic material, the oxidized oils were flowed through tubes packed with definite quantities of finely screened Wilcox sand of a known porosity. In flowing the oxidized oils through the tube a decided plugging of the sand takes place, and the flow of oil is stopped or greatly retarded after only a small quantity of oil has passed through the tube. In this way it has been determined that a satisfactory recovery of oil would be almost impossible if oils in the sand reached a state of oxidation of the same degree as that obtained in the experiments.

The emulsification of oils has been studied also in connection with the use of air as a repressuring medium. While many of these tests in the laboratory have been accelerated by the use of oxygen, the report of the work will treat the subject in such a way that the results may be made to approach

44 Miller, E. C., Migration of Injected Gas Through Oil and Gas Sands of California; Rept. of Investigations 3177, Bureau of Mines, 1932, 29 pp.

average field conditions as nearly as possible. With the issuance of a report on the effect on crude petroleum of air used in repressuring oil fields, the Bureau feels that, for the present at least, it will have satisfactorily answered the most urgent questions regarding changes in the chemical and physical characteristics of crude oil where air is used as a repressuring medium.

Vertical Flow (Including a Study of Pressures and Temperatures in Wells)

The flow of oil, gas, water, or a combination of these fluids takes place because of the differential pressures existing within the system. In solving problems of fluid flow, the petroleum engineer is concerned with two systems, each dependent and exerting a positive effect on the operation of the other. One system includes the space within the reservoir; the other is the vertical column from the face of the sand to the surface of the well.

With this thought in mind, engineers have been working on methods of determining pressures and temperatures in the reservoir for a number of years in order to correlate the fundamental laws of thermodynamics, mechanics and physics, and data obtained in laboratory studies with actual problems encountered in the production of oil from the reservoir and in lifting it to the surface.

In 1928, K. B. Nowels, then supervising engineer of the Bureau's Laramie (Wyo.) field office, recognizing the need for determining the pressures that exist at the bottoms of the wells, began the development of an instrument that would obtain these data which were needed in a study of the energy requirements of vertical flow in wells. It was believed that data on this subject would aid operators in selecting proper sizes of tubing and would make possible the efficient flowing of oil wells with their natural energy and delay the time of exhaustion of the natural energy in the reservoir required to lift the oil to the surface. A second phase of the problem pertains to determinations of the energy requirements for the most efficient flow of oil, gas, and water, or mixtures of these fluids through reservoir sands. Work was inadvertently delayed on this problem through the resignation of Nowels from the Bureau to engage in other work, but during the spring and summer of 1932, a pressure and temperature bomb was developed by C. E. Reistle, Jr., at the Bartlesville (Okla.) station, suitable for the work of the Bureau in studying the thermodynamic and other relationships of vertical flow.

After calibrating the bomb with other instruments developed during the interim period, that had proved satisfactory, a definite study was begun in the East Texas field at the request of operating companies. This work was carried on simultaneously and in conjunction with similarly requested studies of the solubility of gas in the oil under reservoir conditions, to which reference will be made in the following section of this paper.

The Bureau's work on pressures and temperatures in wells in east Texas is different from the periodic "bottom-hole pressure" surveys which have been made by the companies and by the State conservation officers. These organizations have equipment and personnel for making the necessary surveys of pressure conditions which recently have been recognized as an important factor if rational production-control programs are to be established.

The tests of the Bureau include pressure and temperature measurements at various points in the flow strings and at the well heads of selected wells under different rates of flow, both with and without the use of bottom-hole chokes. From these data it is believed that fairly accurate predictions can be made of minimum bottom-hole pressures that will be required to maintain natural flow. These data also should help in solving the problems that will be encountered in the operation of flowing wells in the East Texas field due to the accumulations of paraffin on the face of the sand and the flow columns. A report on "A Study of Subsurface Pressures and Temperatures in Flowing Wells in the East Texas Field and the Application of these Data to Reservoir and Vertical-Flow Problems" was given by Reistle and Hayes before the American Petroleum Institute, Tulsa, Okla., in May 1933, and published in Report of Investigations 3211.

Solubility of Gas in Oil

The industry has long recognized the very important phenomena of the solubility and effects of natural gas and air in crude oils, but it has been the objective of investigators to know whether or not the same straight-line relationships between volume of gas and pressure at which the gas comes out of solution in accordance with Henry's law hold true under reservoir conditions, as has been found in experimental work, where relatively dry gases have been put into solution through the exertion of pressure upon relatively stable samples of crude oil.

To this end, samples were obtained in the Oklahoma City, Kettleman Hills, and Ventura, Calif., oil fields at the casingheads of wells in the pressure range of 1,100 to 1,700 pounds per square inch; after the Bureau had determined by personal discussion or correspondence with representative engineers in all parts of the country that a large majority of these men believed the problem to be one of the most important research projects confronting the petroleum and natural-gas industries.

This inquiry and subsequent developments indicate that there is a growing opinion among engineers and others that the solution of technical and economic problems of the petroleum and natural-gas industries requires the application of fundamental laws and relationships. It is now generally recognized that none of the relationships of fluid flow in reservoirs and in the vertical columns of wells can be thoroughly understood and applied practically without knowledge of pressure and temperature conditions in the wells and the amount and character of naturally dissolved gas in the crude oil and of the conditions which control the liberation of this gas from solution.

From the initial work on casing-head samples of oil-gas mixtures, it was determined that the manner of gas liberation has a definite effect upon the shape of the solubility curves which represent the amounts of gas in solution in the oil at various pressures. Two general types of liberation were recognized - differential liberation and so-called "flash liberation." When gas is liberated from solution differentially, the gases are removed from contact with the oil as rapidly as they are liberated from solution. The essential requirement for "flash liberation" is the presence at the lower pressure (as at the well head) of all the components of gas and oil that are present at the higher pressures (as at the well bottom). The experiments showed that in differential liberation, greater quantities of gas are released from solution per pound pressure drop at low pressures than at high pressures. Such conditions produce solubility curves that are curved, rather than the straight lines in accordance with Henry's law, that were obtained by the earlier investigators who performed their experiments with comparatively dead oil and dry natural gas. In "flash liberation" the gas comes out of solution quite differently from differential liberation, and the solubility graphs resulting from "flash liberation," under conditions where a considerable amount of free or excess gas is present, apparently approach the straight-line relationship in accordance with Henry's law, even though the gas in solution contains appreciable quantities of heavier gases and might be classified as "wet" gas.

The investigators of the problem think that it is probable that differential liberation of gas occurs with respect to the major portion of the oil in a natural underground reservoir, whereas "flash liberation" occurs in the flow string of a well.

These studies of gas liberation also have a direct bearing upon the knowledge of volumetric changes of the oil in the reservoir. It was found that the heavier gases which are liberated at pressures below 100 pounds per square inch absolute, cause much greater shrinkage in the oil and much less reduction of pressure than equal volumes of the lighter gases.

After the data obtained from well-head samples were thoroughly analyzed and conclusions drawn,[45] a "bottom-hole thief," or sampler was constructed for the purpose of obtaining samples at the bottom or at any point in the flow string of the well. The instrument was first given a practical test in the Seminole (Okla.) field, where bottom-hole samples at 400 pounds per square inch pressure were obtained, and more recently has been giving satisfactory service in the East Texas field where reservoir pressures range from 1,400 to 1,500 pounds per square inch.

45 Lindsly, Ben E. Solubility and Liberation of Gas from Natural Oil-Gas Solutions: Tech. Paper 554, Bureau of Mines (in press). (Preliminary report given in Trans. Am. Inst. Min. and Met. Eng., Petrol. Dev. and Tech. 1931, pp. 252-278.)

The present program of field work in east Texas, made possible by the cooperation and financial aid of several oil companies, has been completed. Satisfactory solubility curves for both differential and flash liberation have been obtained at 146°F. and at 90°F. According to the analysis of the data, naturally dissolved gas in the East Texas oil does not begin to come out of solution and expand until the pressure is reduced to 755 pounds per square inch, absolute. Above 755 pounds per square inch absolute, liquid expansion takes place upon a reduction in pressure. Compressibility and shrinkage curves for the oil at the indicated temperatures also have been plotted from the well data.

The more detailed results of this study have been given in a paper by Lindsly entitled "A Study of 'Bottom-hole' Samples of East Texas Crude Oil," before the American Petroleum Institute, in May, 1933, at Tulsa, Okla., and published in Report of Investigations 3212.

Field work in east Texas, on the two related problems of vertical flow -- including pressures and temperatures in oil and gas wells -- and solubility of natural gas in crude oils, had to be suspended for lack of funds after June 30, 1932. Learning that the engineers working on these two problems were to be called back to their headquarters because there was no appropriation to permit them to stay in the field and complete their study (even though the Bureau realized that the results were especially needed at that time in solving problems of rational production-control in the East Texas area) several of the operating companies, who had closely followed the work, took the initiative and devised a plan. Through the generous financial cooperation of 14 or more producing companies, the Bureau engineers were enabled to remain in the East Texas field and finish the tests.

This cooperation of the oil companies, not only in supplying funds to finish these tests, but in making wells available for study, is an outstanding example of cooperative efforts by the Bureau of Mines and oil companies to solve the industry's problems of rational and economical oil production.

Recently, the Bureau began somewhat similar types of solubility and vertical-flow studies in wells of the Oklahoma City field. As conditions will permit, it is hoped that other fields may be studied, thereby making the experimental data more generally applicable throughout the industry and at the same time obtaining specific information on various individual fields that will assist the producers in those fields to operate their wells at maximum efficiency under the conditions imposed by the economic state of the industry.

Summary Statement

This review of some of the major problems pertaining to the production of oil and gas, and related natural-gas transportation problems, in which the Bureau of Mines is actively engaged, indicates the significance of coordinating research in arriving at applied production-engineering methods.

Through these coordinated studies in the laboratory and in the field the fundamental relationships of fluid movement have been found beyond a reasonable doubt to be common to the solution of such problems as the transportation of natural gas through pipe lines, gaging and controlling of natural-gas wells, flow of oil and gas mixtures from the sand face of the wells to the surface, recovery of oil by natural-flow methods wherein proper patterns of well spacing and control of pressures in the reservoir and in the flowing columns are important factors, and recovery of additional oil by imposed stimulative methods of production after reservoirs have been partly depleted. In all of these named practical problems, and others which are met in the production of oil and gas and in their transportation, knowledge is necessary concerning the relationships of the hydrocarbon constituents as affected by pressures and temperatures. In other words, knowledge of the amount of gas naturally dissolved in oil under reservoir conditions and the liberation of this gas from solution upon dimunition of pressure, not only at the surface but in the flowing column and in the reservoir as well, is basicly important in reaching reasonable conclusions regarding this group of correlated production problems, some general results of which have been given.

(3) ENGINEERING FIELD STUDIES

In addition to practical research on problems of oil and gas, the Bureau of Mines conducts engineering studies of typical and important oil and gas fields. These studies are concerned with conditions that influence or control the production of oil and gas. Primarily, they are based upon the correlations of the subsurface formations that constitute the reservoirs and that naturally influence the withdrawal of oil and gas from them.

Detailed studies are made of the logs of the wells in order to find out important information such as suitable landing depths for casing; well penetration into producing zones; the character of the sands and of the oil; water conditions throughout the field; the position of so-called "thief sands;" and many others which may aid economic recovery or hinder operators from obtaining maximum recovery with minimum expense. In addition, the history of a field under study is kept as accurately as well-completion and production records will permit, and methods and equipment at the surface are studied in their relation to subsurface conditions.

These engineering reports on typical fields may be dated from a report issued in September 1919 on the Comanche (Okla.) oil and gas field,[46] although some reports on the Rocky Mountain area were made prior to that time.[47] Since then some 35 reports have been published on fields in Oklahoma, Texas, Louisiana, Arkansas, and the Rocky Mountain region.

Although several field studies are in progress, the Bureau's major activities in this work are now centered in the Oklahoma City pool. This work was started actively in June 1930 after the Bureau made a careful review of the situation to determine the need for such a study.

A report on "The Use of Boiler Feed-Water Heaters with Steam-Powered Rotary Drilling Equipment," by C. E. Reistle, Jr., was published in July 1930 as Report of Investigations 3022, and a paper giving analyses of oils produced in the Oklahoma City field has been issued.[48] A technical paper describing mechanical processes and equipment which were brought out by the industry during the development of the high-pressure sands of the Oklahoma City field is in course of publication.

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46 Swigart, T. E., Underground Problems in the Comanche Oil and Gas Field, Stephens County, Okla.: Bureau of Mines engineering report in cooperation with the State of Oklahoma, Sept. 1919, 42 pp.

47 Tough, F. B., and others, Report of Operations From May 16, 1919 to May 15, 1921: Rocky Mountain Petroleum Association, in cooperation with the U. S. Bureau of Mines, 93 pp. (out of print).

48 Garton, E. L., Analyses of Crude Oils from the Oklahoma City Field, Oklahoma: Rept. of Investigations 3180, Bureau of Mines, 1932, 29 pp.

The issuance of the detailed report on the Oklahoma City field, as it pertains to underground conditions and oil production, has been postponed because of the definite request of the Oklahoma City Chamber of Commerce to prefer a supplemental study of the possible gas reserves in the Oklahoma City field. The gas-reserve report seemed to be more urgently needed than the detailed oil report, and accordingly petroleum and natural-gas engineers of the Dallas office and Bartlesville station, since May 1932 have been concentrating their efforts on this engineering study, which has many economic factors.

A summary report was submitted to the Chamber of Commerce in February 1933, and the detailed "Estimate of the Gas Reserves of the Oklahoma City Oil Fields," comprising 100 manuscript pages, 13 tables, and 15 maps and cross sections, will be available for general distribution and study as soon as it can be printed.

Peg models showing the underground conditions of fields have been a working tool of the Bureau engineers for many years. In the study of the Oklahoma City field, a new type of model showing reservoir conditions from the top of the Oswego limestone to the oil-water contact, has been built. This model is not only unique in design but has been beneficial in giving a visual picture in three dimensions of the contacts of the various formations, the water conditions in the field, and other underground conditions. It is called the "egg crate" model. Sections through the field have been made along east-west and north-south lines of wells. The correlations have been worked out, and the sections showing the logs of the wells have been pasted on one or both sides of stiff cardboard slotted at the proper intervals. When these various sections are fitted together in a manner similar to the construction of an egg crate, a graphic picture of the oil-producing formations is given. Any of the east-west sections can be removed for detailed study without disturbing the rest of the model. The model of the whole field is approximately 14 feet long and 9 feet wide, and has a vertical scale of 100 feet to the inch.

This model has proved to be very helpful in visualizing and understanding the underground conditions of the field.

Prior to the special work on gas reserves of the Oklahoma City field, a preliminary draft of a manuscript dealing with reservoir and subsurface conditions of the Oklahoma City field had been written. Since many of the data were confidential at the time they were furnished by the companies, their release from the companies must be obtained, and other data must be added to bring the report up to date, before the writing can be prepared in final form for publication. Upon several occasions, Bureau of Mines engineers, upon invitation, have attended meetings of engineering committees and have discussed informally with State officials and company engineers technical problems which are of vital concern in the Oklahoma City field.

Among the recently completed field studies is a report on crater wells in the Richland gas field, La.[49] At the instance of the Department of Conservation, State of Louisiana, engineers of the Bureau studied the histories of these seething pits of mud and water through which vast quantities of natural gas were lost for all time. One of the crater wells burned for many months. The report gives data on the cause and repair of crater wells and suggests methods of repair and the importance of preventive measures.

At the completion of a study of the Cotton Valley field,[50] often referred to as an example of a typical ideal textbook structure, a study was begun of the structural features and conditions controlling the accumulation of oil in the chalk and limestone reservoirs of the Zwolle (La.) field.

The Zwolle area is unique in many respects. The limits of the field have not been defined, but the area under development to date is a strip approximately 22 miles long, with an estimated average width of 2 miles. The field now has a maximum width of about 4 miles. The percentage of dry holes is exceedingly high (approximately 60 percent), and many of the commercial wells have been drilled as offsets in the vicinity of the dry holes. In the Zwolle area, acid has been introduced in many wells in an effort to increase the production of oil from the chalk or limestone formations. As in fields of Michigan, this "acid treatment" is reported as being beneficial in stimulating the flow of oil from the producing formations to some of the wells.

With the opening of fields in the great Permian Basin of the West Texas area, attention was drawn particularly to the Roberts-Settles field in Howard and Glasscock Counties. In December 1928, at the request of the Railroad Commission of Texas and the operators, the Bureau gave engineering advice and assistance in a subsurface study of that field, typical of the "lime production" of west Texas. When the work was initiated, it was proposed to prepare a brief report, within a short period, on the water conditions in the 3,000-foot producing zone. Bureau engineers worked with the operators' engineering committee in studying the water problem and in formulating plans to control premature edge water encroachment. A short report, summarizing the results of the investigation and recommending some remedial measures, was submitted to the Railroad Commission in March 1929.

Since the work was begun, other issues affecting the development of this field have arisen. The productive area was extended; new producing horizons have been found; and the entire field has been put under proration. Throughout this transitional period, the collection of additional data on this and other west Texas fields has continued.

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49 Hill, H. B., Crater Wells, Richland Gas Field, La., Tech. Paper 535, Bureau of Mines, 1932, 37 pp. 28 figs.

50 Ross, J. S., Engineering Report of the Cotton Valley Field, Webster Parish, La.: Tech. Paper 504, Bureau of Mines, 1931, 69 pp.

In its effort to complete as many pending problems as possible, the Bureau has assigned one of its engineers to the specific duty of preparing a report for publication which will discuss the Roberts-Settles district as a type field and present outstanding data on the other Permian Basin fields. In the report emphasis is being laid on reservoir conditions, water history, development methods, operating practices, and recovery values, and their economic aspects as applied to limestone reservoirs.

The engineering field reports of the Bureau of Mines not only have a present value, but in the long-time view, they serve as indisputable records, helping in the necessary later repair of wells and in the recovery of oil left in the sand during earlier stages of production. The Bureau's report on the Powell (Tex.) field may be cited as typical of the lasting benefits derived from such studies. Actual work of collecting data and records in this field began in January 1924. Throughout the active producing life of the field, Bureau engineers have been in touch with field operations and have followed in detail the application of remedial measures and the results of the generally adopted "plugging back" program in the field, recommended by the Bureau to control water encroachment and to avert the early menace of flooded wells. In recent months, the published report of the Powell field[51] has been in as great demand as at the time it was issued.

51 Hill, H. B., and Sutton, Chase E., Production and Development Problems in the Powell oil field, Navarro County, Tex.: Bull. 284, Bureau of Mines, 1928, 123 pp.

(4) SPECIAL ENGINEERING PROBLEMS

There is another class of problems, involving certain phases of all the engineering sciences, referred to in the earlier reports of activities of the Bureau of Mines under the heading "engineering technology." Within the petroleum and natural-gas division they are now known as "special engineering problems," although each problem has its own research philosophy. They are undertaken, studied, suspended, or made continuous in the same manner as the refinery, chemistry, and production studies are treated - namely, on the basis of whether or not they justify the effort and money to be expended upon them. This group includes studies of methods of shutting in oil wells, evaporation losses, tank corrosion and internal corrosion of pipe lines, disposal of oil-field brines, hard facing metals for oil-well tools; safety in the petroleum industry, and the compilation of a petroleum bibliography.

Methods of Shutting in Oil Wells

Present economic conditions may be partly responsible for the Bureau's undertaking what was planned as a "short-time" study but which may prove to be the subject for a much longer investigation than was contemplated. This problem deals with methods of temporarily shutting in pumping and flowing wells.

The detrimental effects of water and the losses of oil and gas trapped in sands due to the flooding of productive horizons is now well understood in the industry, and the protection of their correlative rights in a pool has led operators to complete their wells so that losses from this source have been reduced to a very small portion of what they were in former years.

However, production control leading to the need for temporarily shutting in wells after they have been on production for some time, has introduced new factors into this problem, and the shutting-in of wells becomes more than a matter of closing a valve at the casing head or stopping the pump. It is imperative that oil sands exposed in shut-in wells be protected from infiltrations of water from upper and lower water sands, and that the "thief" sands should be sealed to prevent the loss of oil and gas. Very little information is available concerning methods of preparing wells, especially those which have penetrated a series of oil and water sands under different pressures and which have different characteristics, so that these wells may be shut in for a period of time without underground losses.

The Bureau is making an active study of this subject. Filling wells with mud fluid, oil, cement, or a combination of two or more of these materials has been tried, and the experiments are being carefully watched. Experiments are also being made to devise ways whereby these sealing materials may be removed from the wells at a later date without injury to the producing characteristics of the wells.

Prevention of Evaporation Losses

Studies of evaporation losses of petroleum and gasoline were begun in the early years of the petroleum division of the Bureau of Mines. The first work pertained to crude-oil studies, and in sequence losses due to evaporation of gasoline at refineries were studied and reports issued thereon. The more recent studies pertain to methods of reducing evaporation losses at gasoline bulk-storage plants. No complete through-put test at a bulk-storage plant has yet been made, but Report of Investigations 3138,[52] giving the results of tests on standing storage, has helped show the relative magnitude of losses that may be expected from different types of tanks in use at the distributing centers.

Tests have just been completed on two vertical gasoline storage tanks, approximately 11 feet in diameter and 18 feet high, with a capacity of about 12,500 gallons each. These tests were made at Bartlesville, Okla., and will supplement data obtained in previous tests at Kansas City, Mo., reported in Report of Investigations 3138.

The study of evaporation losses appears to be a continuous problem regardless of the fact that through the initial studies of this subject by the Bureau, the industry was made aware of the enormous losses that could be controlled and reduced by the use of properly constructed tank equipment and auxiliary devices. For several years results of the Bureau's studies of evaporation losses have been published only as mimeographed reports, and the Bureau finds itself in a position of no longer having available for distribution many of the reports on evaporation which have been issued. To correct this condition, a bulletin is in preparation which will incorporate the essential findings given in the mimeographed reports and to which will be added many data not contained in them.

Studies of Internal Corrosion

The study of tank corrosion is another engineering problem on which several reports[53] have been issued. Particular interest in this subject developed at the time that fields producing high-sulphur petroleums and hydrogen-sulphide-bearing gases became a factor in the production of crude oil in the United States.

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52 Schmidt, Ludwig, and Wilhelm, C. J., Reduction of Evaporation Losses from Gasoline Bulk Storage Tanks: Rept. of Investigations 3138, Bureau of Mines, 1931, 11 pp.

53 Schmidt, Ludwig, Devine, J. M., and Wilhelm, C. J., The Use of Aluminum for Oil Lease Tanks: Part I - Field Tests; Part II - Laboratory Tests: Rept. of Investigations 3066 and 3131, Bureau of Mines, 1931, 17 and 16 pp., respectively.

Devine, J. M., and Wilhelm, C. J., The Effect of Oxygen on Gaseous Hydrogen Sulphide Corrosion of Tank Steel: Rept. of Investigations 3160, Bureau of Mines, 1932, 19 pp.

From knowledge gained through studies of gaseous corrosion as applied to tanks, it was a logical move to apply these findings to conditions which cause internal corrosion of natural-gas transmission lines. Laboratory tests have been made on various test specimens of materials used in tank and pipe-line construction, and tests have been made in the field to determine what concentrations of hydrogen sulphide and moisture produce the deleterious effects.

In one pipe line, concerning which Bureau engineers were called upon to give the benefit of their experience pertaining to internal corrosion, it appeared at first that the corrosion was occurring at variance with previously established conclusions. Subsequent investigations, however, revealed that the corrosion was not ordinary gaseous corrosion but was a modified form that has been termed "modified gaseous corrosion."

"Modified gaseous corrosion" differs from ordinary gaseous corrosion mainly in that the gas is not only saturated with water vapor but the surface of the subjected metal is also covered by a visible amount of liquid water.[54]

After making some changes in the previous laboratory procedure, it was determined that (1) "modified gaseous corrosion" proceeds as a straight-line function of the pressure that exists on the gas; (2) "modified gaseous corrosion" is a more severe form than ordinary gaseous corrosion, and less hydrogen sulphide is required to promote a severe action; (3) "modified gaseous corrosion" can proceed where there is almost no hydrogen sulphide present because of oxygen, but such corrosion is not great; (4) "modified gaseous corrosion" is more nearly the type of corrosion that occurs under field conditions than the gaseous corrosion previously studied.

As a part of its work on hydrogen sulphide, the Bureau made a survey of producing areas in Illinois, New Mexico, and Texas, and a report[55] was issued on the hydrogen sulphide content of gas in 165 wells, representing 15 oil fields in those States. The percentage of hydrogen sulphide in natural gas produced in various fields ranges from a fraction of 1 percent to as high as 20.5 percent by volume. The extent of fields where hydrogen sulphide is produced with gas and the concentrations encountered clearly indicate that the problems growing out of hydrogen sulphide are of continuing major importance to the petroleum and natural-gas industries.

Disposal of Oil-Field Brines

In many fields as production declines, and particularly where the production is due to an advancing front of salt water, the disposal of brine becomes a serious economic problem to the operators, the owners of property where land

54 Devine, John M., Wilhelm, C. J., and Schmidt, Ludwig., Corrosion of Gases Containing Traces of Hydrogen Sulphide: Tech. Paper 560, Bureau of Mines (in preparation).

55 Devine, J. M., and Wilhelm, C. J., Hydrogen Sulphide Content of the Gas in some Producing Fields: Rept. of Investigations 3128, Bureau of Mines, 1931, 15 pp.

and stream pollution become acute, and to the States. Many oil operators have recognized their responsibility to take care of these wastes, and large nonprofit organizations have been formed in a number of fields in order satisfactorily to take care of an inevitable condition. Producing and refining companies have spent large sums of money to prevent the nuisances, which are in fact economic losses, caused by oil-field brines and refinery wastes.

Some years ago, joining in a Nation-wide study of the problem, the Bureau prepared several reports[56] in reference to contamination of navigable waters and performed an acknowledged service in helping to create a national consciousness regarding this form of preventable waste. More recently particular attention has been given to oil-field brines. At the request of operators and the State of Oklahoma, for example, a study was made of oil-field water-disposal methods. These findings were printed in Report of Investigations 2945.[57] Following this report, disposal methods as practiced in the Salt Flat field of Texas were discussed in Report of Investigations 3059.[58] In this field the highly saline waters are impounded and released to the streams for dilution to unharmful concentrations at periods of high flood. Another report[59] deals with disposal of oil-field brines in the Hendricks oil field.

Many proposals and several attempts have been made to return oil-field brines to underground reservoirs -- presumably in barren formations or down dip from the oil-water contact. Theoretically this practice is logical, but Bureau engineers found that practical considerations should be studied carefully in each area. Some salts will soon clog the pores at the face of the sand of intake wells. Prohibitive pump pressures may develop, and the possibility of contaminating underground sources of potable water or the injury of possible oil and gas producing horizons should be studied carefully and expertly.

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56 Lane, F. W., Bowie, C. P., and Desmond, J. S., Pollution by Oil of the Coast Waters of the United States. Rept. of Investigations 2658, Bureau of Mines, 1924 (out of print).
Report to the Secretary of State by the Interdepartmental Committee, Oil Pollution of Navigable Waters (March 13, 1926, 119 pp.) Appendix 3, General Report of the Bureau of Mines on pollution by oil of the Coast waters of the United States, Jan. 6, 1926, pp. 35-69. Appendix 4, Report of the Bureau of Mines on the action of sea water on fuel oil, Oct. 1925, pp. 70-75.
Lane, F. W., and others, Typical Methods and Devices for Handling Oil-Contaminated Water from Ships and Industrial Plants. Tech. Paper 385, Bureau of Mines, 1926, 64 pp.

57 Schmidt, Ludwig, and Devine, John M., The Disposal of Oil-Field Brines: Rept. of Investigations 2945, Bureau of Mines, 1929, 15 pp.

58 Hill, H. B., Bauserman, E. V. H., and Carpenter, C. B., Development and Production History of the Salt Flat and other Fault Fields of East Texas: Rept. of Investigations 3059, Bureau of Mines, 1931, pp. 30-32.

59 Heithecker, R. E., Some Methods of Separating Oil and Water in West Texas Fields, and the Disposal of Oil-Field Brines in the Hendricks Oil Field, Texas: Rept. of Investigations 3173, Bureau of Mines, 1932, 16 pp.

In the opinion of the Bureau of Mines, every salt-water disposal project is a self-contained problem covered by certain governing principles, the solution of which demands the cooperation of all interested parties.

Attention has been given also to the recovery of commercially valuable salts of iodine, bromine, and of other of the rarer elements in the hope of finding ways of obtaining these salts in such quantities as to convert a present waste into an economic asset.

Hard Facing Metals for Oil-Well Tools

During the past few years rapid development has been made in the drilling of deep wells. Where a few years ago a well that was 5,000 feet deep was considered an outstanding engineering achievement, at the present time depths of 10,000 feet are being obtained. While this increased drilling depth has been due in large measure to improvements in the construction of drilling rigs, rig irons, and other equipment, it has also been made possible by the development of hard-facing metals for oil-well tools. The ability to construct bits that will stand up for relatively long periods of time, when drilling through hard flinty formations, has greatly reduced the time and consequently the cost, of drilling operations. Engineers of the San Francisco office are now collecting data on this subject with particular reference to the types of metals used for hard facing, the methods of applying the material, and the reduced costs of drilling because of their use.

Safety Engineering in the Petroleum Industry

One of the primary reasons for the organization of the Bureau of Mines was to reduce the number and severity of accidents in the mineral industries. In the petroleum and natural-gas division, accident-prevention work has been carried on actively as an integral part of its other engineering studies. The Bureau has long maintained that accident prevention should be dealt with on a scientific basis, and although only two engineers of the Bureau are definitely assigned to studies of accident prevention in the petroleum industry at any one time, the individual work of these two men does not adequately represent the Bureau's activities in studying methods to prevent personal injury and loss of property. The experience of the whole Bureau of Mines stands back of the active work and technical writing of these two engineers.

A complete article in itself would be required to outline the activities of the Bureau of Mines on petroleum safety. Exclusive of special articles, some 80 reports dealing with petroleum safety and health, including fires and explosions, have been published by the Bureau and are available to the public. Some of these writings date back to 1913, but in 1923 two petroleum engineers were assigned to work on safety problems in order that they might collect and analyze statistical data on accidents and with this knowledge, theretofore unavailable, make engineering studies where the highest frequency and severity rates existed. Drilling and production problems were the first to be studied

with the view of suggesting remedies for unsafe conditions. Natural-gasoline plants have had their share of attention, and a report has been published.[60]

Some recent technical work on safety problems deals with methods of reducing the hazards in the design and operation of petroleum cracking equipment at refineries, where the dangers from failure are greatly increased because of the corrosive effects of sulphur compounds (or erosive action) on equipment containing petroleum vapors that are subjected to high degrees of temperature and high-working pressure. A report is in preparation on this subject.[61]

When the hydrogen sulphide problem presented itself in west Texas and fields of the Permian Basin, the Bureau sent engineers and chemists to those areas in order to advise the oil companies and the men in their working places regarding the toxicity of this gas, with which so few were familiar, and to suggest remedial measures that would lessen the number of accidents from exposure to this gas, which in its poisonous effects is comparable with hydrogen cyanide. A report[62] was quickly prepared and followed by its sequel,[63] and both papers were given wide distribution throughout the industry.

Since the hydrogen sulphide studies of 1926 and 1927, a considerable amount of progress has been made, and methods have been developed by the industry and others that had not been recognized or that had not been proved at the time of reporting the earlier work. In order that there may be a general reference to the subject of hydrogen-sulphide poisoning and methods of preventing it in all departments of the petroleum and natural-gas industries, the Bureau has in preparation a technical paper which will bring this subject up to date.

One of the laboratory investigations on this work pertains to the pyrophoric characteristics of certain products of corrosion by hydrogen sulphide that have been known to ignite spontaneously and that present a hazard about which very little authentic information so far has been published.

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60 Shea, G. B., Safety at Natural-gasoline Plants: Tech. Paper 462, Bureau of Mines, 1929, 109 pp.

61 Marek, R. L., Safety at Petroleum Cracking Plants: Tech. Paper 551, Bureau of Mines (in press).

62 Yant, W. P., and Fowler, H. C., Hydrogen Sulphide Poisoning in the Texas Panhandle, Big Lake, Tex., and McCamey, Tex., Oil Fields: Rept. of Investigations 2776, Bureau of Mines, 1926 (out of print).

63 Fowler, H. C., Prevention of Hydrogen Sulphide Poisoning in Handling and Refining High-Sulphur Petroleums: Rept. of Investigations 2847, Bureau of Mines, 1927 (out of print).

The Bureau began giving first-aid instruction to the petroleum industry in 1920. At present there is hardly a person employed in the petroleum and natural-gas industries who is not familiar with the care of injured persons in case of an emergency. Not all of these men actually have been trained by the Bureau of Mines because the task of actually giving the instruction has become too great. The Bureau has sometimes been criticized for not making more trainers available, but funds for this work are definitely limited. If key men in any organization receive instruction in the application of first-aid methods, the company should be in a position to carry on its own first-aid instruction under the supervision of the Bureau of Mines, whose trainers hold examinations for Bureau of Mines first-aid certificates. First-aid training has been a large factor in accident-prevention work. The relationship of these two subjects is so thoroughly understood that it need not be elaborated upon here.

Petroleum Bibliography

The Bureau has compiled a bibliography of petroleum and allied substances since 1915. At first these bibliographies were published as annual bulletins of the Bureau of Mines. The amount of technical writing on oil and gas subjects has expanded rapidly, and the task of abstracting the articles became too great for one bibliographer to handle. Accordingly, through the cooperation of the Special Libraries Association and the endorsement of the American Petroleum Institute, this work was put on a cooperative basis as of January 1929.

Since that time "Recent Articles on Petroleum and Allied Substances" has been published monthly at the San Francisco petroleum field office in the form of 50 or more mimeographed pages. Until recently the bibliography has been mailed without charge to anyone requesting the service. The list contained approximately 560 names. However, in accordance with recent legislation, this publication has been put on a subscription basis, and the money derived therefrom is placed in the United States Treasury and not applied on the cost of printing. It appears that under existing conditions, the Bureau cannot long continue the publication of this work, in spite of the splendid assistance given by various technical librarians of oil companies and others.

This bibliography, placing in available form the carefully prepared abstracts of substantially all articles published in this and foreign countries on petroleum and allied substances, should be continued under some plan, but because of lack of funds, the Bureau feels that the cost of maintenance of this service must be assumed by the industry, or the work stopped.

Availability of Bureau of Mines Data on Petroleum and Natural Gas

Upon occasion the statement has been made that the Bureau holds back too many of its data at times when there is greatest need for them. The Bureau has endeavored always to make the information on a given subject available as soon as that information is properly authenticated within its own organization, provided no other circumstance is a controlling factor -- as, for example, certain confidential requirements. (Information made available to the Bureau of Mines from an outside source and specified "confidential" is meticulously treated as such until the donor of the information in question removes the confidential restriction.)

The Bureau of Mines is necessarily conservative in this matter because it realizes that too frequently conclusions are drawn without substantiating proof, and the general practice is followed that it is better to withhold a publication until the facts are established than to issue reports suggesting certain possibilities which, upon further investigation, may have to be modified. Wherever conditions will permit, especially where data are informative, rather than basic in nature, "short-time" reports are issued, interspersed between the longer reports on major projects.

CONCLUSION

The studies of the United States Bureau of Mines on petroleum chemistry and refining, production of oil and gas, pipe-line transportation of natural gas, engineering field studies, and special engineering problems, concerning which some general results have been given in this paper, by no means make up a complete list of technical research projects on oil and natural gas. Some years ago the Bureau was asked to compile a list of petroleum and natural-gas problems on which information was needed by the industry and on which the Bureau, in line with its function as a Federal research agency, could do work advantageously if funds were available. The number of problems which finally remained on the list after careful scrutiny was 105. No doubt this number would be increased if a similar tabulation were made today.

However, it is not the objective of the Bureau of Mines to see how many problems it might study, but rather to put into usable form the best obtainable information on the principal oil and gas problems, which, as a Federal agency, it is peculiarly fitted to study and make reports on for the improvement of methods of rational conservation of oil and gas and for the betterment of the social economy of the country.

The statement has been made that research in the oil and gas industry has been carried so far that the knowledge now available is acting as a deterrent rather than a help toward stabilization of economic conditions. It is true that engineering advancement in general has added to the complexity of many economic problems, but the statements regarding the detrimental effects of research are made by persons who fail to recognize the fact that good engineering is synonomous with sound economic principles.

Industry, largely through its own efforts, has come to accept the fundamentals of oil and gas conservation and safe operation - suggested, endorsed, and worked for by the Bureau of Mines since its organization. The judiciary of the country is coming to consider the conservation of oil and gas and rational methods of preventing waste, both physical and economic, in a different light than it did in the earlier years of the industry. The public at times has been inclined to criticize industry and perhaps has failed to understand the significance of technical research.

The true public interest is broader than the knowledge of the present retail price of gasoline. The public must know more about the fundamental concepts of recovery and utilization of oil and gas - not concerning itself so much with the detailed findings and interpretations of specific data - but it must appreciate how oil and gas and its orderly development and economic distribution to the consumer affect each individual socially and economically. The public must know, for example, something of the relationships of the oil-gas-energy attributes of a reservoir as they affect the correlative rights of all parties interested in the land. It must have some knowledge, also, of why it is necessary for technicians to study the characteristics of motor

fuels and lubricants in order that more suitable grades of these products may be made available to meet the requirements of its automobiles, even though the actual technique is obscure to many individuals. It must recognize that national physical wastes attending production and manufacture cannot be changed into national assets in a short space of time.

Oil and gas conservation and its effect upon the social economy are extremely complex. The action of no one organization or group of organizations can cope with the situation. Progress results from the combined efforts of all forward-looking groups who are trying honestly to reach a definite objective.

Well-informed persons who have had an opportunity to study the work of the Bureau of Mines have stated that the bureau has selected types of work on general problems which the industry itself usually is not in a position to do, either in its own company research organizations, commercial research organizations, or in schools and colleges. They have stated also that the work of the Bureau, although under different conditions, is continuing as it did in former years to exercise a very definite influence in preventing waste and in conserving the natural resources of oil and gas.

The Bureau of Mines as a Federal agency has maintained its fore rank position as a research organization, studying oil and gas problems, through carefully guarding against premature judgment; its findings are based on fact, insofar as pertinent facts can be determined through experimentation and investigation; its conclusions are given without prejudice of property lines or viewpoint engendered by commercial affiliation. After severe criticism within the Bureau, most of the manuscripts are sent to nationally recognized authorities in the oil and gas industries for critical review and suggestion regarding presentation of the facts. Upon final approval and release, the findings on any subject are made equally available to large and small companies, individual operators, regulatory agencies, landowners, and the public.

With these precepts before it, and shunning mediocrity, the Bureau of Mines has been able, in the face of an increasingly complex social economy, to continue its proper function of developing the philosophy of research projects of national scope that pertain to oil and gas.

I. C. 6738

July 1933.

Department of Commerce

United States Bureau of Mines
Scott Turner, Director

INFORMATION CIRCULAR

BLASTING PRACTICES AS THEY AFFECT THE ROOF OF COAL MINES IN OHIO, PENNSYLVANIA, AND WEST VIRGINIA

BY

J. N. Geyer

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

BLASTING PRACTICES AS THEY AFFECT THE ROOF OF COAL MINES IN OHIO, PENNSYLVANIA, AND WEST VIRGINIA[1]

By J. N. Geyer[2]

REASON FOR INVESTIGATION

In coal mining it is essential to maintain the roof over working places and roadways in the safest condition possible. To this end every effort should be made to win the coal with the least injury to the immediate roof strata, as broken strata certainly present a greater hazard than a sound roof. The chief causes of injury to the roof in mining are inherent in the method of development and recovery and the blasting practices followed. The first, which may produce severe stresses in the roof, weakening it over comparatively large areas, has been discussed in previous papers[3] and will not be treated here. The second cause is local in effect, but with the same general method of blasting being employed throughout a mine having uniform roof conditions, similar effects on the roof may be expected.

In the Bureau's investigation of accidents caused by falls of roof and coal it has been necessary to make a careful study of blasting practices and their effect on the immediate roof. The purpose of this paper is to show the effect of the observed methods of blasting on different types of roof and to point out methods that have been proved to be least harmful to the roof strata.

The data used in preparing this paper have been compiled from reports made by Bureau engineers engaged in the study of falls of roof and coal, and embrace mines opened in the Pittsburgh coal in Ohio and Pennsylvania and the Pittsburgh and Sewickley coals in West Virginia.

DESCRIPTION OF COAL BEDS AND ROOF

Pittsburgh Bed

The Pittsburgh coal bed varies from 52 to 108 inches in thickness in the mines studied. Aside from the variations in thickness, the physical characteristics of the bed as observed in the different mines have a marked similarity. The coal is of bituminous rank, of hard texture, and is characterized by its bright appearance and blocky fractures. Two bone or shale partings are generally present in the bed, the upper one occurring 24 to 48 inches below the roof and the second 3 to 6 inches below the first. These partings are 1/4 to 1 inch thick.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used; "Reprinted from U.S. Bureau of Mines Information Circular 6738."

2 Associate mining engineer, U.S. Bureau of Mines, Pittsburgh Experiment Station, Pittsburgh, Pa.

3 Geyer, J. N., Factors and Conditions That Aid in Alinement of Pillar Extraction Lines in Coal Mines; Inf. Circ. 6727, Bureau of Mines. In press.

Paul, J. W., and Plein, L. N., Methods of Development and Pillar Extraction in Mining the Pittsburgh Coal Bed in Pennsylvania, West Virginia, and Ohio; Bureau of Mines Information Circular. In press.

The immediate roof is not as uniform in character as the coal bed, but it has certain members persistent over a large part of the field. Chief of these is the jointed gray clay known as "draw slate" or stone that occurs immediately over the coal. This varies in thickness from little more than 1 inch to over 4 feet and shows wide variations over short distances; the average thickness over the field is approximately 12 inches. The draw slate is hard when first exposed, but it disintegrates rapidly on exposure to the atmosphere and becomes plastic in the presence of water. As the clay is rarely self-supporting and because of its tendency to disintegrate when exposed, timber support is impracticable except as a temporary measure. This necessitates taking the draw slate or else leaving a stratum of head coal below it strong enough to hold the draw slate between timber supports and thick enough to protect it from contact with the mine air.

Overlying the draw slate is a stratum of high-ash coal which in a few places is over 3 feet thick but is generally between 10 and 18 inches and averages about 12.5 inches. The rider coal is not as persistent or as uniform as the draw slate. In some localities it is absent or appears as a carbonaceous shale; in other sections, the coal is multiple bedded, the benches being separated by shale or clay partings, 2 to 5 inches thick. Where the draw slate is taken down, the rider coal generally affords a dependable roof if it is at least 6 inches thick. The members of the immediate roof overlying the rider coal consist of clay and shale of varying strength and thickness. These are often important factors in roof control and support, but they seldom require special consideration in blasting, except as they may be affected by the underlying strata.

Sewickley Bed

The mines studied operating in the Sewickley coal were in Monongalia County, West Virginia. In these, the coal varied from 45 to 72 inches in thickness. The coal is of bituminous rank, hard texture, and has a blocky fracture. The bed often has a bone or shale parting 1/4 inch to 1-1/2 inches thick near the middle, separating it into two benches.

The immediate roof consists of shales having a maximum thickness of 10 feet but sometimes disappearing entirely, in which case a hard sandstone lies immediately on the coal. The immediate roof shales are generally strong, but in some parts of the field the roof behaves as if it were heavily stressed, causing the strata immediately over the coal to break with little or no warning in headings and rooms. No definite indications were found, however, between the roof behavior and the method of blasting.

STATE BLASTING REGULATIONS

The blasting practices in any coal bed are regulated to a large extent by the mining laws of the State in which the mines operate. Most of the laws relating to blasting are primarily to prevent blown-out shots; however, these same measures tend to mitigate the violence against the roof. The sections of the bituminous coal mining laws of Pennsylvania, Ohio, and West Virginia relating to blasting practices that may effect the roof follow:

Pennsylvania Bituminous Mining Laws, 1926

Article IV

Section 9. The mine foreman shall direct that the coal is properly mined before it is blasted. "Properly mined" shall mean that the coal shall be undercut, center-cut, top-cut, or sheared by pick or machine, and in any case the

undercutting shall be as deep as the holes are laid. In mines generating explosive gas in quantities sufficient to be detected by an approved safety lamp, when the coal seam is 5 feet 6 inches or more in thickness, "properly mined" shall mean that in all entries less than 10 feet wide, wherein the coal is undercut, it shall also be sheared on one side as deep as the undercutting before any holes are charged and fired, or the coal shall be blasted in sections by placing the first hole near the center of the coal seam. . . . . .

Section 10. . . . .The mine foreman, or the assistant mine foreman under instructions from the mine foreman, shall direct that the holes for blasting shall be properly placed, and shall designate the angle and depth of holes, which shall not be deeper than the undercutting, center-cutting, top-cutting, or shearing, and the maximum quantity of explosives required for each hole, and the method of charging and tamping. . . . .

Section 14. In such portions of a mine where explosive gas is being generated in quantities sufficient to be detected by an approved safety lamp, and in which locked safety lamps are used, the mine foreman shall employ a sufficient number of competent persons, who are able to speak the English language, to act as shot-firers, whose duty shall be to charge, tamp, and fire all holes properly placed by the miners, and to refuse to charge any holes not properly placed. No holes shall be fired by any person other than a shot-firer. . . . .

In all mines in which coal is blasted from the solid, all holes shall be fired when all the workmen are out of the mine except the shot-firers and other persons delegated by the mine foreman to safeguard property.

Article XI

Section 8. Shot firing by electricity.

89. Shot firers: Only competent persons, who have the necessary training and skill, and who have been properly instructed in the work, and duly authorized by the mine foreman, shall be allowed to fire shots electrically in any mine.

Article XXV

Special Rules. Duties of Miners

Rule One. . . . .It shall be the duty of the miner to mine his coal properly before blasting, and to set sprags under the coal while undermining, to secure it from falling. . . . .

West Virginia Mining Law, 1929

Section 23. In any mine in which solid shooting is done the district mine inspector is authorized to prescribe the condition under which solid shooting may be done; any operator or mine foreman, who causes or permits any solid shooting to be done therein without first having obtained a written permit from the district inspector, or any miner therein who shoots coal from the solid without first having obtained permission so to do from the operator or mine foreman, shall be guilty of a misdemeanor and upon conviction shall be fined as hereinafter provided.

Section 62. . . . .No person shall fire more than one shot at a time, and after firing said shot he shall not return to the working place until the smoke has cleared away; . . .

Section 63. In no case shall more than one kind of explosive be used in the same drill hole, and every blasting hole shall be tamped, except as is necessary to accomplish cushion blasting, full from the explosive to the mouth, and no coal dust or inflammable material shall be used for tamping. Cushion blasting shall not be allowed in any case unless written permission is granted by the department of mines. Dynamite shall not be used in blasting coal. No fuses shall be used unless permission is granted by the mine foreman and in no case shall fuses be used of less length than the drill hole.

Rules

Use of Explosives

Rule 9. Not more than one shot shall be fired at one time and only one kind of explosive used in the same hole.

Rule 11. Shooting off the solid is prohibited, except by permission in writing from the mine foreman, under conditions prescribed by the District Mine Inspector.

Ohio Mining Law, 1931

Section 110. Squibs and fuses; missed shots. . . .Two or more shots shall not be fired in the same face, unless all fuses used are at least 12 inches different in length, or unless the shots are fired simultaneously by electric caps or electric squibs.

Section 113. Regulation of drill holes. When a miner drills holes in the face of coal for the purpose of blasting and such drill hole bottoms in the permanent roof, he shall not charge or fire such drill hole. In no case shall a miner drill a new drill hole in such a way that any part of it shall be within 12 inches of any part of an unused drill hole. . . . . .

Section 115. Solid shooting prohibited without permit. The owner, lessee or agent of any mine shall not order or permit solid shooting in a mine in this State unless he has obtained written permission to do so from the chief, division of mines, who may issue such permit when in his judgment such solid shooting is necessary for the just and reasonably profitable operation of such mine.

It will be noted that certain provisions are made in each State for blasting coal from the solid. In Ohio, definite provision is made against charging and firing shots in holes drilled into the permanent roof, and in Pennsylvania the angle depth and maximum charge of explosive used in each hole is to be designated by the foreman. In West Virginia, the law stipulates that shots shall be fired singly and that the workmen shall not return to the face until the smoke has cleared away.

Methods Used in Pittsburgh Bed

Mines in the Pittsburgh coal may be divided into two groups - those in which the draw slate is taken down and those in which the draw slate is supported. In the former group the entire coal bed is mined and the draw slate because of its inherent weakness and slaking characteristics when exposed must be taken down. In the latter group the coal bed is thick enough to allow leaving several inches of head coal to protect the draw slate without hinder-

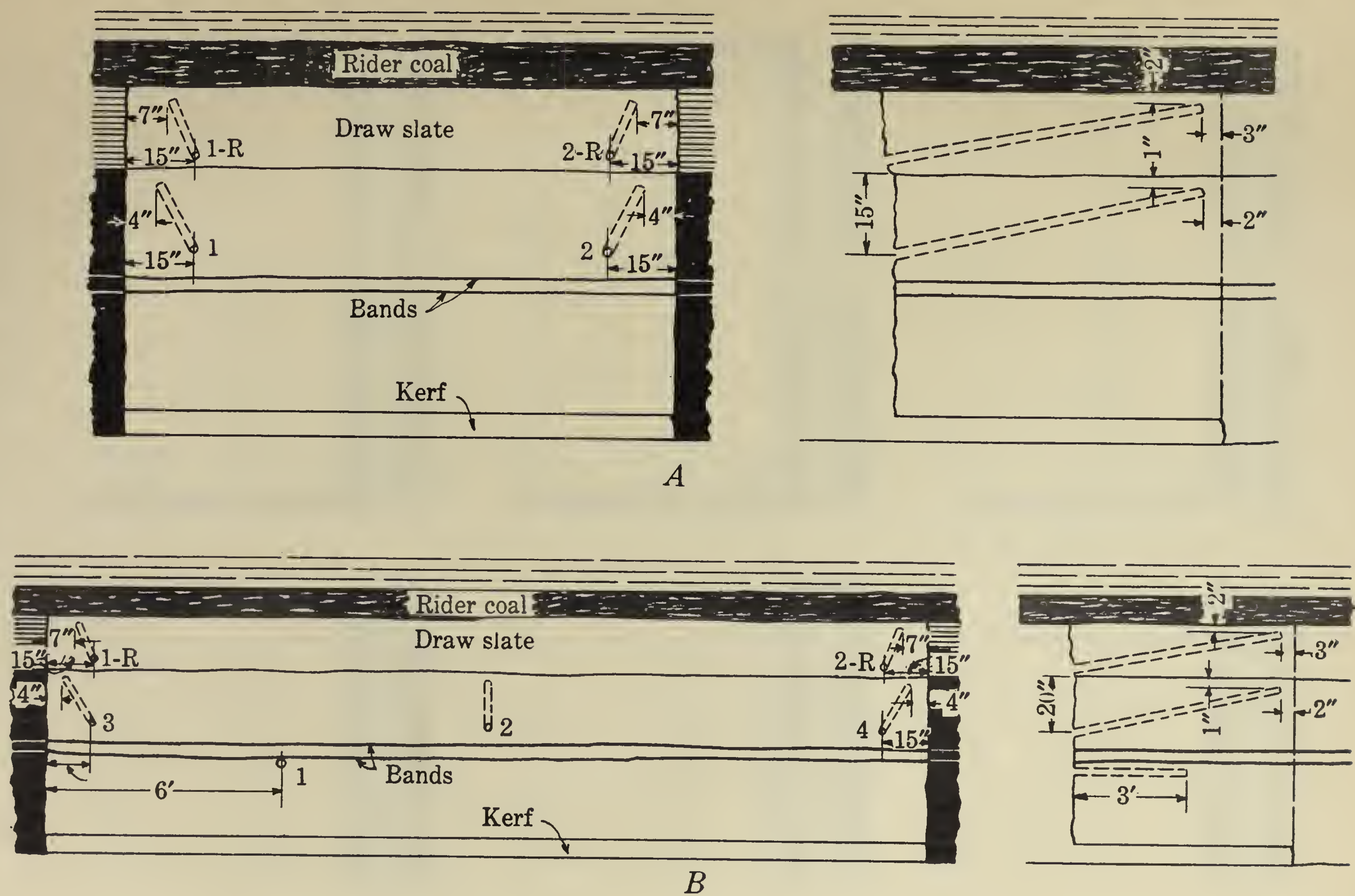

Figure 1.—Position of drill holes in Pittsburgh coal bed where draw slate is taken down. *A*, Heading face and section; *B*, room face and section.

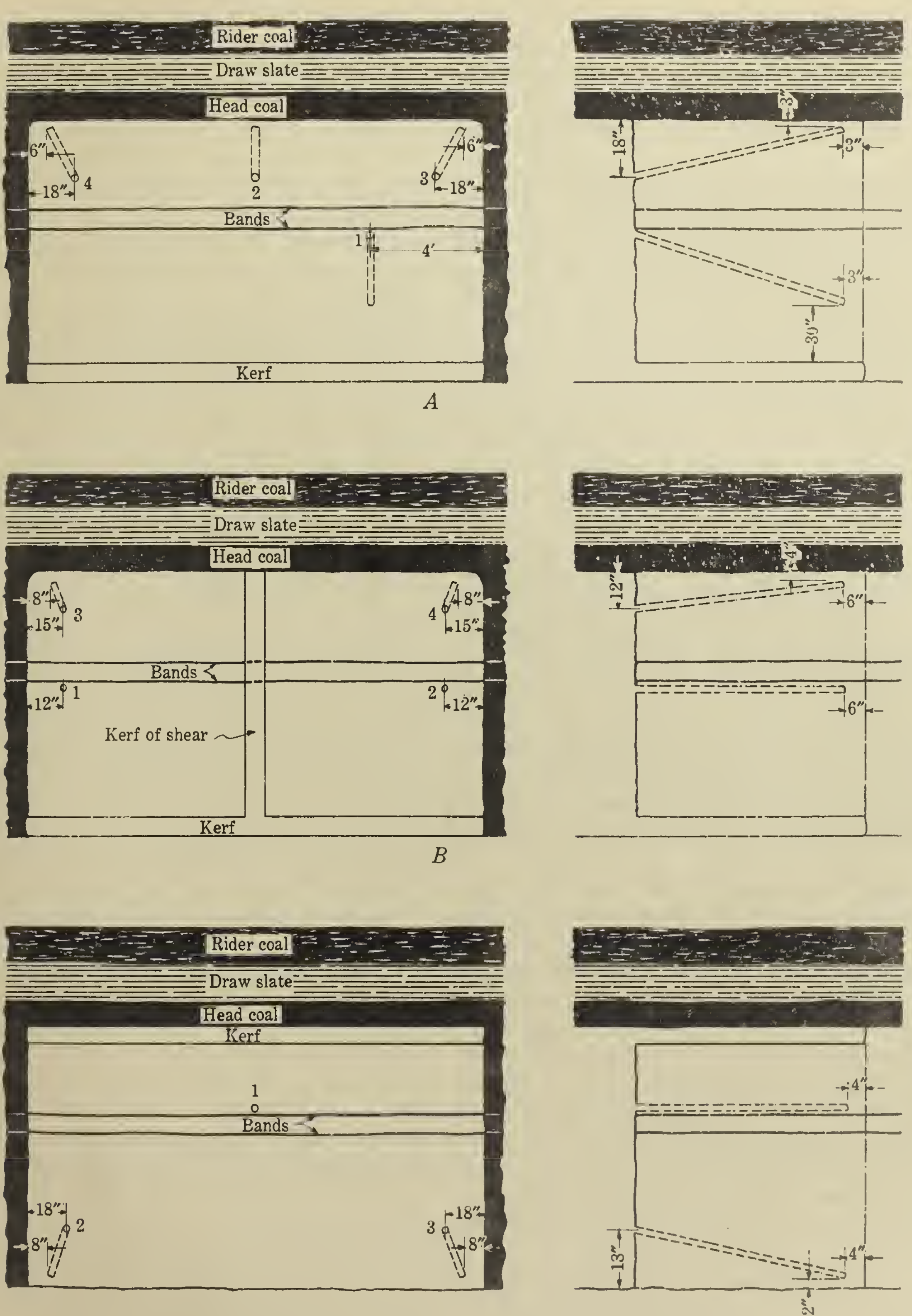

Figure 2.—Position of drill holes in Pittsburgh coal bed where head coal is left to protect draw slate. *A*, Where coal is undercut and hand loaded; *B*, undercut and center sheared, hand or machine loading; *C*, where coal is top-cut.

ing transportation. If this head coal is of good quality, it is then recovered as the pillars are extracted; or, if of poor quality, as is often the case, the coal is not recovered.

In approximately 60 percent of the mines studied in the three States (all in Ohio, 67 percent in Pennsylvania, and 30 percent in West Virginia) the draw slate was taken as the places advanced.

Cutting or Mining.- In Ohio, all coal was cut by shortwall or chain-breast machines driven by electricity, the kerf being 5-1/2 to 6 feet deep and at, or a few inches above, the bottom of the bed. In the Pennsylvania mines, where the draw slate is taken, all rooms, headings, and crosscuts are cut by machines and over 50 percent of the pillar coal is machine cut. The machines were electrically driven and of the shortwall type except in four mines, in one of which a turret-type machine that remained on the track was used and in the other three of which combination cutting and shearing machines were used. The kerf was on or within a few inches of the floor in each case. In the three mines, where machines equipped for shearing were used, the coal was center sheared to the back of the undercut. This was done in each case to facilitate blasting for machine loading. The only mines in West Virginia in which the draw slate was taken were in the Panhandle district. In these all of the coal was bottom cut with shortwall-type electric machines.

Top-cutting machines were used in 30 percent of the mines in which head coal was left to protect the draw slate. In the other mines the coal was cut on the bottom with chain-breast, shortwall, and track-mounted machines, with the exception of two mines in Pennsylvania where the coal was mined by pick. The coal in two Pennsylvania mines and in parts of two mines in West Virginia was center-sheared to the depth of the undercut. The depth of kerf in machine-cut places varied from 5-1/2 to 8 feet and, where the coal was hand mined, the kerf was about 4-1/2 feet deep. In the mines where pick mining was used exclusively, the cuts were made about 4 inches below the draw slate.

Drilling.- Holes for blasting were drilled with power drills in one mine studied in Ohio, 11 in Pennsylvania, and in 5 mines and part of another in West Virginia. In most instances the drills were mounted on the cutting machines. Compressed air operated the drills in two mines; in all others the drills were driven by electricity. Breast augers were generally used by the miners for hand drilling.

Headings are driven 8 to 10 feet wide in the mines where the draw slate is taken, and 2 holes are drilled for blasting the coal. If the draw slate does not fall with the coal or is too firm to be barred down, 1 or 2 holes are drilled in the rock to blast it down. Rooms where the draw slate is taken are generally 18 to 24 feet wide and 2 to 4 holes are drilled in the coal; 1 to 3 holes are drilled in the draw slate if it cannot be barred down. Figure 1 shows the positions of holes in headings and rooms where the draw slate is taken. The dimensions shown are averages for the district. The holes drilled in the draw slate are started near the bottom of the stratum and not at a fixed distance from the roof. Hole 1 or the snubbing shot is used in only a few mines; it is placed at any point between the rib and center holes and is generally just below the bands. The top holes in nearly all of the mines are stopped against or within 2 inches of the draw slate. This sometimes shatters the rock and in any event would tend to weaken it. In a few mines the holes were stopped 4 to 6 inches below the slate. This practice brought the coal down effectively without disturbing the roof.

In mines where head coal is left to protect the draw slate the headings are driven 10 to 12 feet wide, the usual width being 12 feet. Rooms in these mines are 10 to 16 feet wide, 12 and 14 feet being common widths. Where the coal is undercut and blasted for hand loading, 2 to 4 holes are drilled; the general positions for these holes are shown in figure 2,A. Holes corresponding to 3 and 4 are always drilled; these are started 6 to 30 inches from the ribs and 8 to 36 inches from the head coal. They terminate 1 to 12 inches from the ribs,

not more than 6 inches from the head coal and within 6 inches of the back of the kerf. Hole 2 is seldom used unless the coal does not break away from the head coal readily; it is generally parallel to the line of sights and has the same pitch and depth as the rib holes. Hole 1 is drilled in several mines; it is generally started below the bands but in a few instances is just above or between them and may be at any point between the rib and the center of the place. The hole may be drilled flat or may dip toward the back and should terminate within 6 inches of the back of the cut.

Figure 2,B, shows the general position of holes where the coal is undercut and center-sheared. As in figure 2,A, holes 3 and 4 are always drilled but holes 1 and 2 are not generally used except with mechanical loading. The latter holes are drilled flat and parallel to the line of sights. In some mines where mechanical loading is used it has been found that shearing produces lumps too large to be handled by the loading machine conveyor without jambing against the roof. In one of these mines this was overcome by drilling 6 holes and not shearing. The 2 additional holes were drilled in the middle, one in line with the roof holes and one below the bands. The 3 holes below the bands were then drilled parallel with the line of sights, but dipped so that the back was only 18 inches above the kerf. This causes the coal blasted down by these shots to be thrown forward, thus making more room for the coal brought down by the top shots.

Top-cutting machines were used in several mines, principally when the head coal was left up. About half of these mines used 3 holes, while in the others only 2 were drilled. Figure 2,C, shows the common practice for drilling in top-cut places. Holes 2 and 3 have the general positions shown in all of the mines. The top hole when used is always in the center with respect to the ribs and is drilled parallel to the line of sights. In most instances it is started at the bands and is parallel to the bedding planes as shown; however, in one mine this hole is started 36 inches above the floor and is pitched to be approximately 14 inches from the kerf at the back; the hole is stopped 6 inches from the back of the cut. This position is used because the shots are all fired before the loaders enter the mine, and by drilling the hole in this position the coal loosened by the shot is thrown from the face, allowing greater freedom for the bottom shots.

Explosives.- Permissible explosive and black blasting powder in pellet and granular forms were used in the coal and in the draw slate where it was blasted. Permissible types of explosive were used in most of the mines, including those listed as nongassy. The quantity of permissible explosive used in each drill hole varied from 1 to 5 cartridges and of pellet powder from 1/2 to 3 cartridges. The coal produced per pound of explosive ranged between 4.3 and 18.1 tons for permissible explosive and 4.5 to 15.8 tons for black blasting powder in both forms. Shot firers were employed in 60 percent of the mines, and in a few mines either all or a part of the shooting was done on the preparation shift before the loaders entered the mine. In the other mines all shots were fired by the loaders, using squibs for black powder and electric detonators for the permissible explosive.

Effect on the Roof.- Where the draw slate was taken, more carelessness was noted in placing the top holes and in the quantity of explosive used than in the mines where head coal was left. In some cases the loaders placed the back of the top holes against the roof purposely to break the draw slate without blasting in it. This sometimes produced the desired result, but more often the blast was not sufficient to bring the slate down, yet would loosen it enough to make its support while loading the coal difficult and extremely hazardous. Observations in a large number of mines have shown that the explosive need not be less than 4 inches from the roof to break the coal from the draw slate and that at this distance there is little danger of injuring the roof.

In mines leaving head coal to protect the draw slate, more care is generally exercised in drilling and charging the holes to prevent shattering the head coal. However, in a few

mines the head coal was repeatedly shattered or cut along the ribs by the force of the explosive. This was particularly noticeable in one mine where the holes were drilled with power drills, swivel-mounted, on top of the mining machines, the latter remaining on the track while cutting and drilling. No provision was made for lateral movement of the drills, nor could they be raised or lowered except by tilting the drill; consequently, all holes radiated from a common point and were frequently drilled too high into the head coal or extended too far into the ribs. With the exception of this mine, the head coal appeared to be less broken in mines where the holes were drilled by special crews using power drills than where this work was performed by the loaders.

Methods Used in Sewickley Bed

Head coal was left to support the immediate roof in only one of the mines studied in the Sewickley bed; in all others the coal was taken to the immediate roof. Headings and crosscuts are driven 12 to 14 feet wide and rooms are 14 to 20 feet wide, the average being 18 feet.

The coal was undercut on the floor with electric shortwall machines in all but one mine where the coal was hand mined. The coal is cut to a depth of about 6 feet with machines and about 4-1/2 feet when hand mined.

The general practice in the district is to drill two holes for blasting in headings and rooms. All holes are drilled by the miners, using breast augers. The positions of drill holes shown in figure 3 are typical for all mines except the one leaving head coal. In this mine the holes are started about 24 inches below the roof and pitched so that the hole terminates over the back of the kerf about 12 inches below the roof. The coal then breaks when blasted about 6 inches above the hole, leaving 6 inches of firm coal against the roof.

The coal was blasted with permissible explosives in nearly 60 percent of the mines and black blasting powder in either pellet or granular form in the other mines; the loaders fired all shots at any time during the working shift. The coal produced per pound of permissible explosive ranged between 7.9 and 11.8 tons and with black powder 7.3 and 10.0 tons. The permissible explosive was detonated electrically, current being taken from small nonpermissible dry cell batteries, and the black powder, in both forms, was fired with squibs.

In the Sewickley bed no definite proof of blasting's having seriously affected the roof was found. In some instances, however, where the roof was jointed or heavily stressed, blasting may have opened the joints or weakened the roof, causing larger falls than otherwise would have occurred.

METHOD OF PROTECTING THE ROOF

In general, three factors, separately or in combination, affect the safety of the roof in blasting: The position of the kerf; the position of the drill holes; and the characteristics and quantity of explosive used. These may be manipulated separately or in combination to produce the desired results. Changing the position of the kerf from the bottom to the top or some intermediate point might afford the greatest protection to the roof, but in many instances this would entail considerable expense to the company in replacing the machines in use with a type designed for top cutting; so it is often advisable to consider the less expensive methods first.

Choosing the Correct Explosive

It is now possible to obtain a permissible explosive that meets any reasonable condition encountered in coal mining, provided that the explosive is handled in the manner recommended

by the manufacturer. Choosing the most satisfactory explosive may require considerable experimental work by an engineer who is experienced in handling explosives and has a working knowledge of the types of permissibles on the market. Most manufacturers of explosives have engineers who will study conditions in a mine and recommend the explosive manufactured by their company that will give the best results. The final choice should be the explosive that leaves the coal in the most marketable form with the least possible injury to the roof.

Position of Drill Holes

A factor of as great and sometimes greater importance to the roof than the explosive is the position of the drill holes, because the most satisfactory explosive obtainable will fail of its purpose if fired in a hole improperly placed. Some of the more common errors observed in the mines studied are:

(1) Drilling holes too deep for the depth of cut;
(2) Directing the holes toward the ribs;
(3) Inclining the holes at too great an angle toward the roof;
(4) Holes too close to the roof;
(5) Drilling the holes before the place has been cut.

The first three of these errors result in or may produce effects comparable to blasting off the solid. This shatters the coal, producing an excessive amount of slack which is often a drug on the market, and shatters or loosens the roof strata, causing needless expenditure in cleaning falls or setting additional supports to make the weakened roof safe. A frequent cause of drilling the holes too deep is that the miner assumes that the place has been cut to the full depth of the cutter bar and he then drills the holes to the full depth that the machine would be able to cut; consequently, if a cutter bar were 6 feet long, and, due to irregularities on the face, had cut only 5 feet deep, the hole would be 1 foot in solid coal; and, if three cartridges 8 inches long were charged in the hole, one half of the charge would be in solid coal. This may easily be prevented if the miner will measure the depth of the cut at the point where the hole is to be drilled. The hole should then be stopped 4 to 6 inches short of the depth of the kerf.

Directing the hole toward the rib, or gripping the rib, is common practice among miners. This is probably due to the fact that machinemen frequently cut a fan-shaped kerf whose rib lines the miners are inclined to follow in drilling. Either gripping the ribs or pitching the holes at too great an angle with the roof will produce effects similar to blasting off the solid, as illustrated in figure 4, in which the shaded portions show the coal subject to the greatest shattering effect. It will be noted that practically the entire force acting perpendicular to the drill hole may be directed against solid coal, which confines the gases and causes excessive shock to the coal and roof. Both of these practices may be corrected easily by the supervisory staff in the mine and should effect a saving to the miner in explosives, produce less fine coal, and reduce the cost of maintaining the roof. Rib holes should be drilled parallel to the line of sights, even if the cutting machine has gripped the ribs. The distance from the rib at which the holes should be drilled depends on the character of the coal, thickness of the bed and width of the places, but they should be drilled at the maximum distance at which the shots will trim the ribs. In the Pittsburgh coal bed the distance ranged between 6 and 30 inches, but tests conducted by the Bureau of Mines indicate that 12 inches will generally give best results. All top holes should be drilled as nearly flat as possible and in the Pittsburgh bed the back of the hole should be at least 4 inches from the roof. Where the miner uses a breast auger in high coal, a bench should be provided for him to stand on while drilling so that the hole may be kept level instead of slanting upward. The holes should be stopped approximately 6 inches short of the kerf and in no event should holes be drilled deeper than the kerf.

Drilling holes before the place has been cut is a practice sometimes encountered. This practice is most common in mines where the loader is required to wait on cars or where the cutting is done while the loader is in the mine. In either instance the loader, wishing to waste no time, will drill holes for the next cut as soon as he has removed the loose coal from the face. Holes so drilled may be deeper than the cut or may penetrate into the rib beyond the cut; in either case the shot will be fired in solid coal.

Position of the Kerf

In some mines having an exceptionally tender roof, little improvement can be effected by changing the explosive or the position of the drill holes. Under such conditions it is generally advisable, and sometimes imperative, that top-cutting machines be used; or, if the nature of the bed makes top cutting impracticable, the cut may be made at a point near the top, and the coal above the kerf may then be barred down or blasted with light charges of explosive. A roof study was made in one mine opened in the Pittsburgh bed where it would have been economically impossible to have operated the mine without top cutting. Overlying the coal in this mine was a calcareous clayey shale interspersed with lenticular masses of limestone which could not be held when exposed. To protect this roof, the coal was top cut 6 to 12 inches below the roof and the coal above the kerf was not recovered. In another mine, room pillars that had been standing for several years were being recovered by pick work because of the weakened roof. The miners cut a kerf 8 to 12 inches below the draw slate and it was noted that the head coal in these pillar places was stronger than the head coal in advance places which were cut on the floor with shortwall machines. In the latter places the explosive had not shattered the head coal, but when tested it emitted a drummy sound, and the vibrations set up indicated that the coal had been loosened along the cleats and bedding planes.

Supervisory and Disciplinary Measures

Little can be accomplished toward protecting the roof without close supervision and rigid discipline. Tonnage men naturally wish to produce the greatest amount of coal possible with the least effort. The machinemen, therefore, try to cut as much coal as possible, and the loaders desire to blast the coal in a manner that will reduce pick work to the minimum. As a result, unless closely supervised and disciplined, the machinemen will cut the working places too wide and the loaders will overcharge the holes, thus shattering the coal and injuring the roof. One of the first duties of an operating company is to draw up a code of rules and standards for the guidance of the officials and employees. Those relating to blasting should embody the best practices for the production of marketable coal with the least damage to the roof and should set forth clearly all operations that may be standardized, including the manner in which the loader should prepare the face for cutting, instructions for the machinemen, position of drill holes and manner in which holes are to be charged and fired. The officials should then supervise the work closely and tolerate no violations of the rules or standards. It is often difficult for a section foreman to give each phase of the work on his section the time it requires. Where shot firers are employed they have authority to refuse to fire any shots in places where the company rules or State mining laws are violated. If competent shot firers are employed, this method provides closer supervision than can be had when loaders fire the shots, and precludes the possibility of shots' being fired in violation of the rules. A most commendable policy of some companies is to employ as shot firers only men who are certified first-grade mine foremen.

Where shot firers are not employed, the damage caused by improper blasting is often done before it comes to the attention of the officials. The method of disciplining those having violated the rules is generally a reprimand for the first offense, a lay-off for the second, and dismissal for the third. This policy, however, works an unnecessary hardship on the company; for the first offense, it must bear the cost of timbering the place if cut too wide or if the roof has been broken; and, for the second offense, it loses the productive efforts of the violator in addition to the expense of making the place safe. One company has overcome a part of these difficulties by requiring the violator of the rules to make the place safe on his own time as punishment for the first and second offenses, and, after the third, he is dismissed. This rule applies to both machinemen and loaders, depending on whom the responsibility for the violation rests in the opinion of the official.

Because of the greater centralization of responsibility, the use of power drills operated by regular drill crews will result generally in a more uniform placement of holes than can be accomplished when the drilling is done by the individual miner.

ADVANTAGES GAINED BY IMPROVED BLASTING PRACTICES

The benefits to be derived from improving the cutting, drilling, and blasting practices include increased safety to those underground and economy to the miner and operator. The more important advantages, stated briefly, are:

(1) Protection to the roof;
(2) Increased marketability of the coal;
(3) Economy in handling waste material at the face;
(4) Facilitated separation of impurities from the coal.

Protecting the roof from all unnecessary shock often prevents joints from being opened which will destroy any keying effect in the strata that might otherwise have held or assisted in holding the roof. Open joints also provide a passageway for air and moisture which cause the disintegration of many strata that occur over coal beds. A roof so weakened must then be securely supported or the loose material taken down and the overlying strata frequently supported, thus increasing the cost of the coal. Such a roof creates a serious hazard to those working under it, especially haulagemen, who may be injured by falls caused by derailed cars striking timbers carrying loose rock. The compensation costs of such accidents have not been segregated at the mines studied, but it is an item of cost that should be given careful consideration. In passing, it is suggested that if all coal companies would keep as detailed a description of the cause or causes of each accident as are kept of the nature and extent of the injury, they would find it less difficult to formulate constructive measures for the reduction of accidents.

Practically any blasting practice adopted for the purpose of protecting the roof should increase the marketability of the coal by reducing the quantity of slack made. Some companies report no reduction where the coal is cut horizontally and sheared, but this they attribute to the increase in machine cuttings and not to greater degradation as a result of the explosive. In top cutting no reduction in fine coal will be effected unless the drill holes are placed to reduce shattering of the coal as described for top holes; however, if the holes are drilled flat or nearly so and parallel to the line of sights, the reduction in fine coal should be similar to that in undercut places.

Blasting the coal with a minimum of degradation will generally result in a cleaner product at the face, especially if the impurities occur as bands which are readily shattered. When the refuse is badly shattered, it may escape the notice of the loader or he may not take time to remove all the small pieces, whereas large pieces could seldom fail to attract his attention. Removing the refuse at the face would reduce the amount of cleaning neces-

sary at the tipple and obviate payment for refuse at the price of coal. Some companies have saved the cost of elaborate cleaning plants by cutting out a parting or making the cut immediately above or below the band and removing the impurity before blasting. The practicability of this should be carefully considered because large saving may possibly be made in this manner.

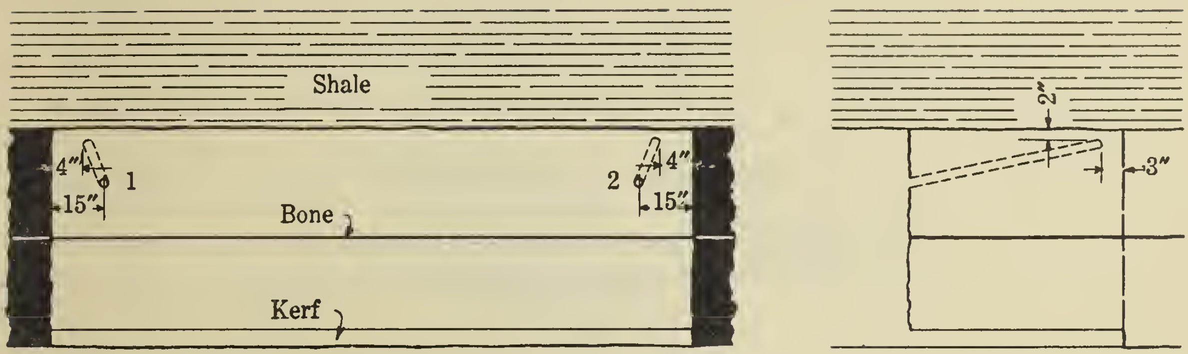

Figure 3.—Position of drill holes in Sewickley coal, West Virginia.

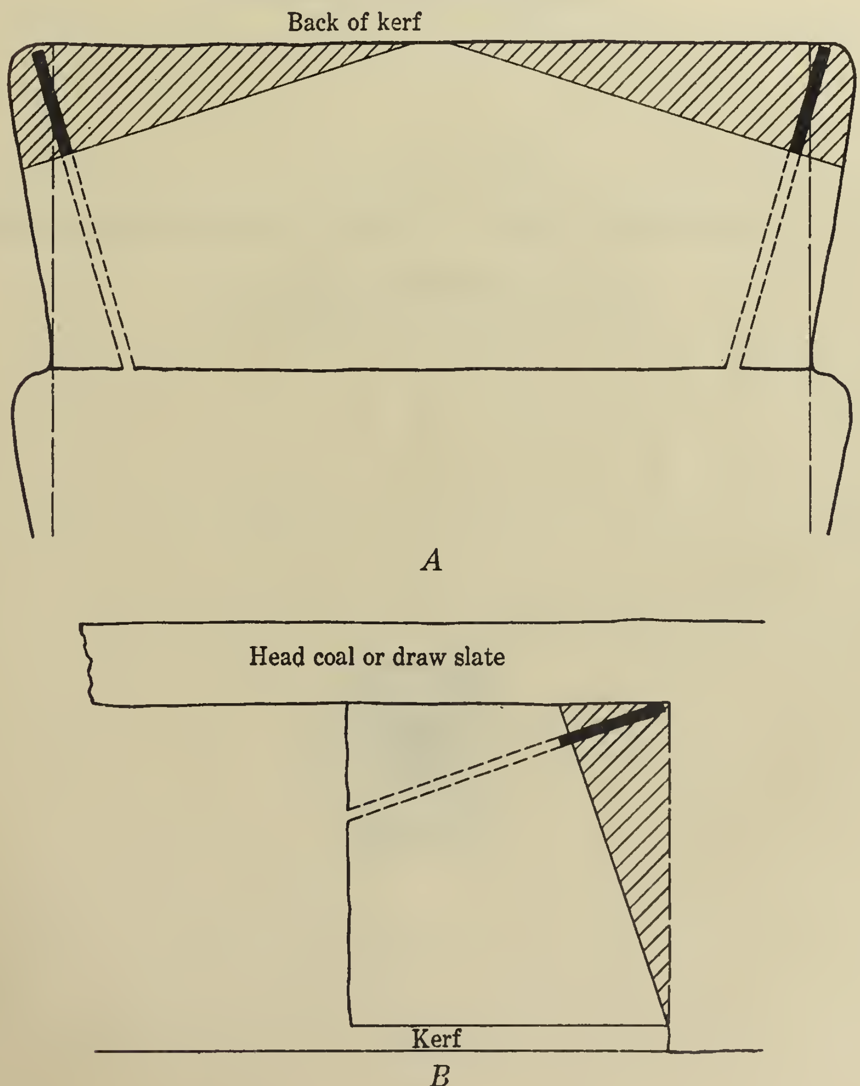

Figure 4.—Areas in which explosive acts against solid coal when the holes grip the ribs and are at an angle with the roof. *A*, Plan; *B*, side elevation.

I. C. 6739 July, 1933

Department of Commerce
United States Bureau of Mines
Scott Turner, Director

INFORMATION CIRCULAR

MILLING METHODS AND COSTS AT THE GOLDEN CYCLE MILL, COLORADO SPRINGS, COLO.

BY

L. S. Harner

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

MILLING METHODS AND COSTS AT THE GOLDEN CYCLE MILL, COLORADO SPRINGS, COLO.[1]

By L. S. Harner[2]

INTRODUCTION

This paper which describes the milling practice at the Golden Cycle mill is one of a series of similar papers being prepared by the United States Bureau of Mines.

The mill of the Golden Cycle Corporation, formerly known as the Golden Cycle Mining and Reduction Co., is located at Colorado Springs, Colo., and has been operated since 1907. It is now and has been from its inception almost exclusively a custom mill.

HISTORY

The original mill had capacity to treat 37,000 tons of oxidized silicious ores per month by roasting and cyanide methods. When it treated what may be termed "basic ores" the capacity was 32,000 tons per month.

A small-capacity cyanide unit was added to the original plant about 2 years ago for the purpose of treating miscellaneous gold-silver ores which did not require roasting. In the latter part of 1929 a concentrator unit was added for the treatment of complex sulphide ores by selective flotation methods. The tailings of this latter unit are further treated in the cyanide plant for additional recoveries of gold and silver.

The changes in operating methods at the Golden Cycle plant since 1907 have been made because of--

(a) Additional operating knowledge gained from direct experience and from information received from outside sources.

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1 The Bureau of Mines will welcome reprinting of this article provided the following footnote acknowledgment is made: "Reprinted from U. S. Bureau of Mines Information Circular 6739."

2 Manager, Golden Cycle Corporation, and one of the consulting engineers, U. S. Bureau of Mines.

(b) Mechanical improvements which include materials of better quality furnished by manufacturing companies.

(c) Changes in methods and equipment brought about by changes in the character of ores treated.

The development of milling methods at this plant cannot be shown by cost and recovery figures over the entire period of plant operation. The milling practices that yielded high recoveries and low costs when the plant started operations would result in low extractions and very high costs if practiced on the present type of ore treated. As the character of ore changed, new methods and new expensive and more efficient equipment were necessary if operations were to continue. At all times changes were preceded by experimental work.

ORE TREATED

The gold ores which are treated by roasting and cyanide methods are the sulphotelluride ores received almost exclusively from the Cripple Creek district. These ores are chiefly gold bearing and contain very small amounts of silver and negligible quantities of base metals such as lead, copper, zinc, arsenic, antimony, and mercury.

Gold and silver ores treated by direct cyanide methods are received in small shipments from quite widely scattered districts.

Complex sulphide ores which are treated by selective flotation methods are furnished in large part by the well-known old mining districts of Clear Creek and Gilpin Counties, the Alma, Leadville, Creede and San Juan areas.

The Cripple Creek ores received by the mill are classified either as silicious or basic ores; the former are obtained from the upper levels or so-called "oxidized" and "leached" zones of the mines and the latter from below the oxidized zones. The siliceous oxidized ores can be readily crushed, ground, and roasted, whereas the denser, harder, and slicker ores obtained below the oxidized zone give more difficulty in the crushing, grinding, and roasting operations. The tabulation which follows presents typical analyses of these two classes of ore.

Typical analyses of Cripple Creek ores milled

Constituents, percent	Siliceous oxidized ore	Dense hard ore
Insoluble	86.70	75.90
Al_2O_3	2.30	3.40
Fe	3.50	4.00
CaO	1.57	5.12
S	1.79	1.8 to 2.30
MgO	0.50	1.40
Loss ignition	3.20	6.50

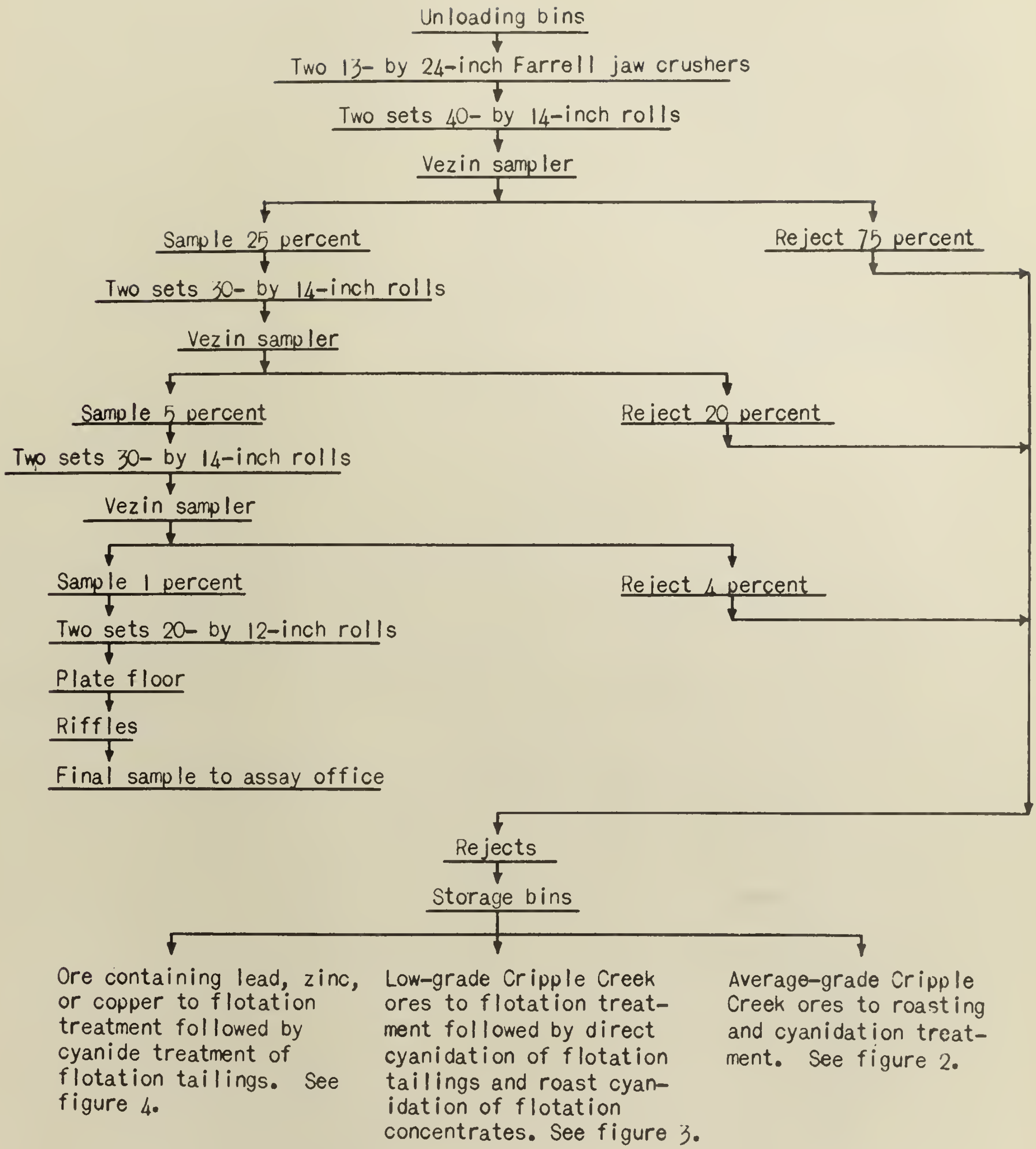

Figure 1.- Flow sheet of sampling mill.

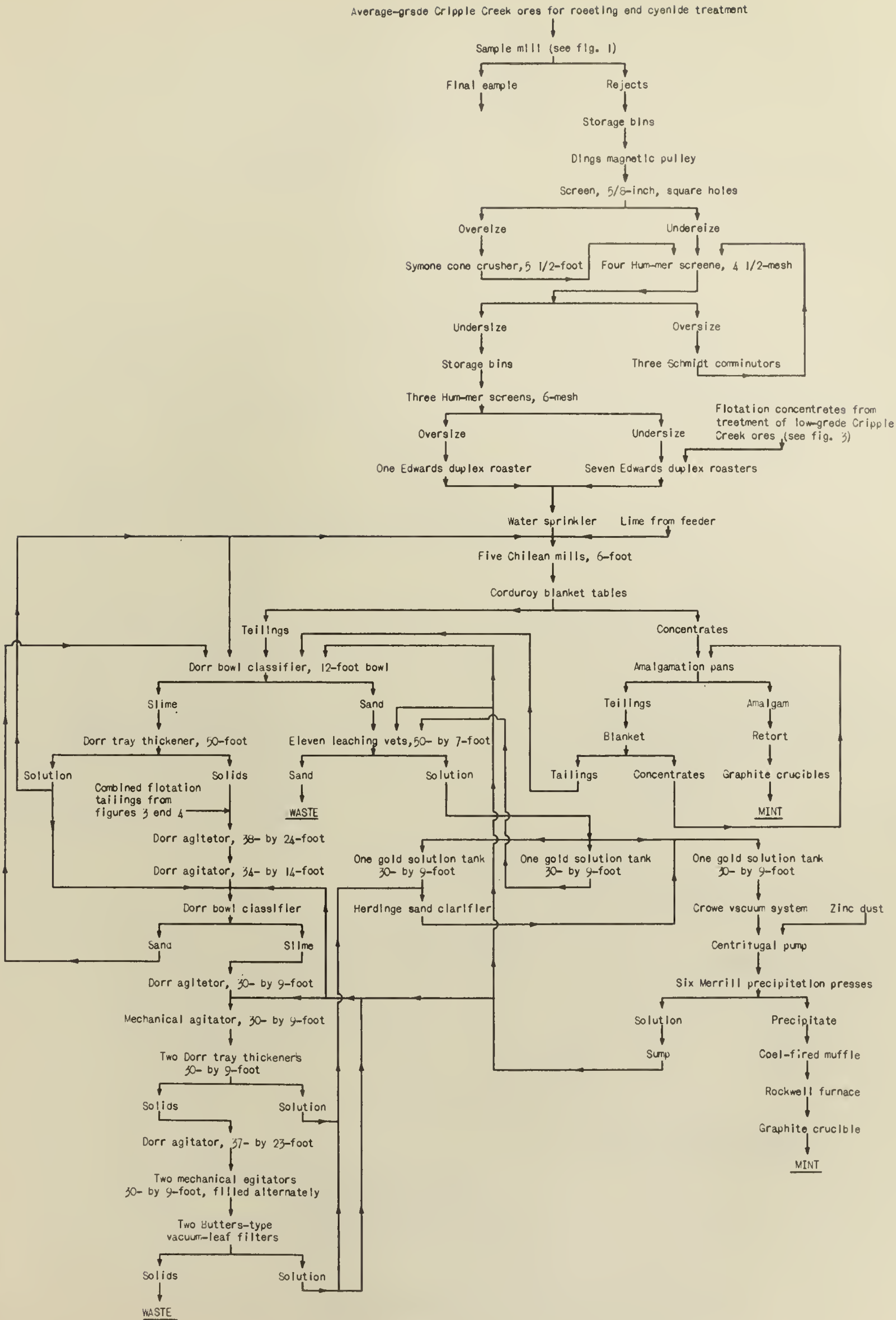

Figure 2.- Flow sheet of treatment for average-grade Cripple Creek ores.

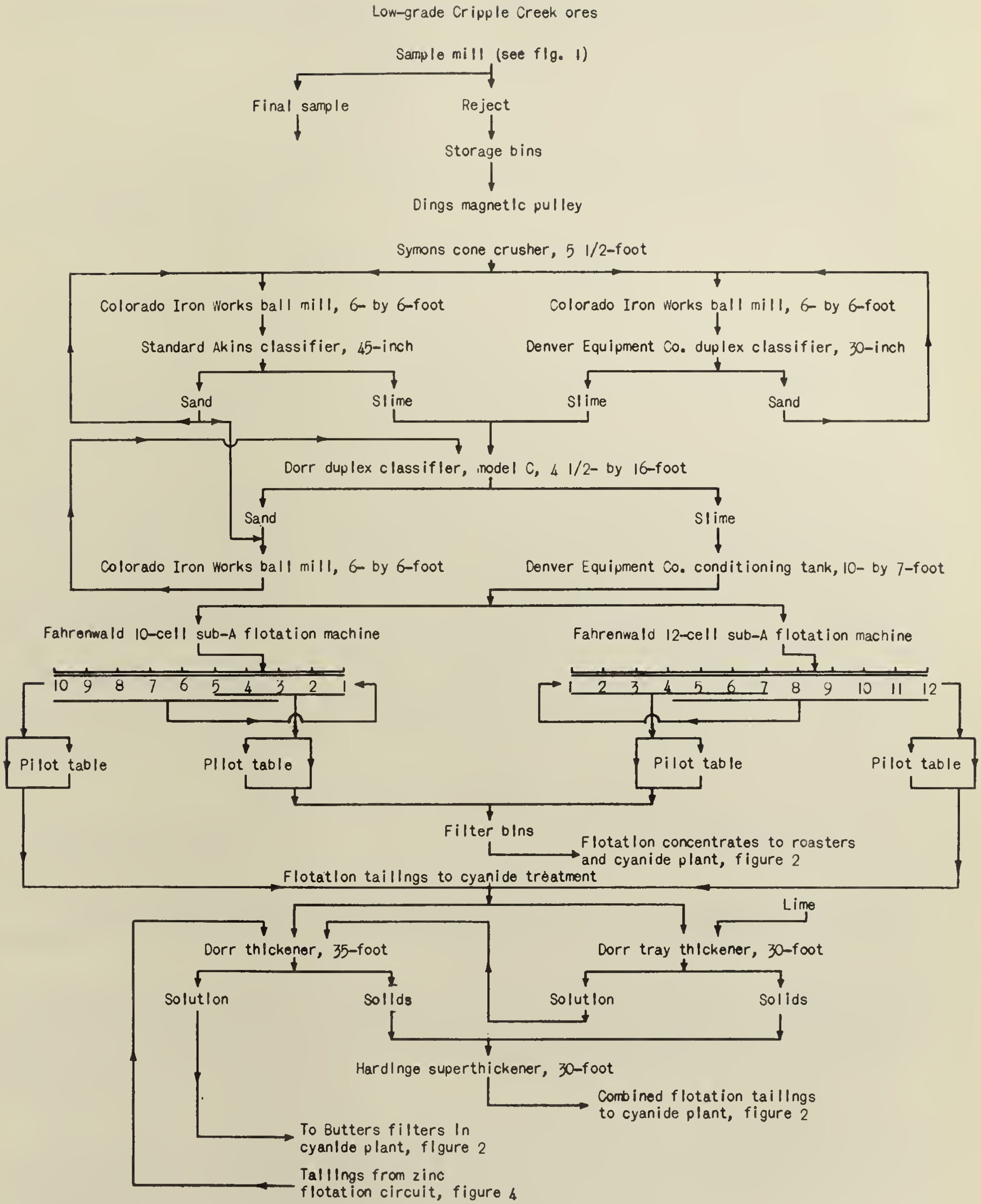

Figure 3.- Flow sheet of treatment for low-grade Cripple Creek ores.

The slight physical and chemical changes that take place in the ores after mining and before milling have no appreciable effect on ore treatment.

WATER AND POWER

Water is pumped from a near-by creek and is stored in seven large tanks.

Power is chiefly supplied from the company's steam electrical plant which is located at the mill. The city of Colorado Springs furnishes any additional power required.

PRESENT METHODS OF TREATMENT

Sampling

As previously mentioned, the Golden Cycle plant operates chiefly on custom ores and is therefore equipped with a complete sampling unit. Figure 1 shows the flow sheet of the sampling plant.

From the unloading bins the ore is fed to two 13- by 24-inch Farrell jaw crushers. The crushed product is further reduced by two sets of 40- by 14-inch rolls and then passed to a Vezin sample cutter which takes a 25 percent cut. The sample is again reduced by two sets of 30- by 14-inch rolls and then passes to a second Vezin sample cutter which removes a 20 percent cut. The sample is further reduced by 30- by 14-inch rolls and passed to a third Vezin sampler which takes a 20 percent cut. This sample which amounts to 1 percent of the original material is crushed by two sets of 20- by 12-inch rolls and further reduced in size by riffles. The final sample is dried, ground, and sent to the assay office. The rejects from sampling operations, which constitute the plant feed, are conveyed to storage bins.

Classification of Treatment Methods

The plant as previously noted treats three general classes of ores by the methods which follow:

1. The average-grade Cripple Creek gold ores are treated by roasting followed by cyanidation. Figure 2 presents a flow sheet of this treatment.

2. Low-grade Cripple Creek gold ores are first treated by flotation as shown in figure 3. The flotation concentrates are sent to the roasters which treat the average-grade Cripple Creek ores, and the roaster calcines follow the treatment outlined in figure 2. The flotation tailings are cyanided directly as shown in figure 3.

3. Ores which contain lead, copper, or zinc are treated by selective flotation methods as indicated in figure 4. The lead concentrates and zinc concentrates produced are shipped to smelting points; the flotation tailings of the zinc circuit are cyanided directly as shown in figure 3.

This paper does not describe the details of flotation treatment as practiced at the Golden Cycle plant but deals mainly with the roasting and cyanide treatment presented in figure 2.

TREATMENT OF AVERAGE-GRADE CRIPPLE CREEK ORES

Referring to figure 2, the ore is conveyed from the storage bins on a 14-inch belt which is equipped with a Ding's magnetic head pulley for the removal of tramp iron. The conveyor belt discharges the material onto a stationary screen having 5/8-inch holes and set at a slope of about 40°. The oversize passes to a 5 1/2-foot Symons cone crusher set at 3/8 inch; the crushed product joins the stationary screen undersize and both are fed to four Hum-mer screens equipped with 4 1/2-mesh Rek-Tang screen cloth. The Hum-mer screen oversize is further reduced by three Schmidt comminuters; the latter are dry-grinding ball mills. These mills operate in closed circuit with the 4 1/2-mesh screens. The undersize products of the 4 1/2-mesh Hum-mer screens comprise the feed to the roasters and are conveyed to storage bins.

The Symons cone crusher will handle 100 tons of ore per hour, the feed being 3 1/2- to 4-inch maximum size and the product 1/2-inch maximum size.

Ores received in the early operating days of the Golden Cycle plant could be reduced to roasting size quite readily. However, as lower levels in the mine were reached the character of the ore changed to a more basic variety and at times to somewhat dolomitic unoxidized material. The crushing of these lower-level ores became more difficult.

At the beginning of operations the crusher product was reduced to roaster feed size by three 6- by 6-foot Schmidt comminuters. The latter used 5-inch balls and were equipped with diagonally slotted screens having 9/64- by 1/2-inch holes. These screens were attached to each mill and operated in closed circuit with it, the screen oversize being returned to the mill.

As the difficulty of crushing ore increased a 36-inch Symons horizontal disk crusher and a 48-inch Symons vertical disk crusher were added to the crushing circuit ahead of the Schmidt ball mills. The remodeled unit operated in a satisfactory manner for a while, but as more of the mines reached deeper levels the cost of crushing ore increased.

The next change made was the addition of a 10-foot by 48-inch dry-grinding Hardinge ball mill to the grinding units already installed. This change was followed by the installation of a 5 1/2-foot Symons cone crusher.

The Symons cone crusher displaced the horizontal and vertical Symons disk machines and also rendered the operation of the 10-foot Hardinge mill unnecessary. Since the installation of the cone crusher, power costs in this department have been reduced one half and the time required for crushing has been reduced one third. The saving made by this installation amounts to $0.08 per ton of ore handled.

Roasting

The ore is delivered from the roaster storage bins to three Hum-mer screens equipped with 6-mesh Rek-Tang screen cloth. Table 1 gives a screen analysis of the feed to the Hum-mer screens.

The screen oversize is roasted in one Edwards duplex furnace;[3] the screen undersize, with concentrates from the flotation unit handling low-grade Cripple Creek ores, is distributed to seven Edwards duplex roasters. Each roaster will handle from 115 to 130 tons of ore per 24 hours. Screen analyses of coarse and fine roaster feeds are presented in table 1.

The Edwards roasting furnaces are 115 feet long by 13 feet wide and each has an active hearth area of 1,495 square feet. A cooling hearth, 44 feet long and 13 feet wide, adjoins each roaster at the discharge end. Roaster hearths have a slope of 2 inches per foot.

Each roaster is equipped with 54 revolving rabbles which move the ore from the feed to the discharge end of the furnace. The rabble arms of the roasting hearth revolve at a speed of 3 r.p.m. and those of the cooling hearth at 6 r.p.m. Each roaster requires 16 hp. for operation; ore passes through the roaster in the average time of 6 hours.

The rabble arms of the hearths are water cooled. Cooling water used per roaster amounts to 90 to 110 gallons per minute depending on the temperature of the hearth and the rate of feeding ore. The water enters the rabble arms at a temperature of from 23° to 33° C. and after leaving the rabbles, is sent to a cooling pond and from there is pumped to storage tanks for reuse.

Colorado Springs lignite coal of the following analysis is used for roaster fuel:

Water	per cent	20.20
Volatile matter	do.	54.65
Fixed carbon	do.	19.75
Ash	do.	5.10
Sulphur	do.	0.30
Heating value	B.t.u. per pound	8,700

The fire box in use is of the Western Fire Box Co. type; it is a semigas producer and uses live steam. The consumption of coal in the producer at present ranges from 230 to 240 pounds per ton of ore roasted, and an additional 8 percent is required to generate the steam for the producer. Sizing of the ore before roasting has been found to reduce the coal required from 75 to 100 pounds per ton of ore when the roasters operate at a temperature of from 800° to 900°C. Lower roasting temperatures are used at times and under these conditions the saving of coal by sizing the ore is not so pronounced.

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3 Hofman, H. O., General Metallurgy: McGraw-Hill Book Co., New York, 1st ed., 1913, p. 424.

Extensive tests to determine roaster-stack losses were made in 1909. Gas velocities and gas pressures were determined and dust samples were caught in filter bags. At the beginning of these tests the loss of metal in stack-dust was found to be $0.13 per ton of ore. This loss was reduced to $0.04 per ton of ore by lowering the feed discharge point into the roasters, by providing better draft control and by choking the calcine discharge.

Temperatures are controlled by Brown indicating pyrometers; recording pyrometers are also in use for plotting temperature curves over long periods. Sulphur analyses are made each day.

The roasting of the basic ores may result in the production of two very undesirable compounds; namely, CaS and $CaSO_4$.

When CaS is present in the roaster calcine it consumes a large amount of cyanide in the leaching operation; it is also a reducing agent, and acting in this manner decreases the dissolution rate of the gold and at times decreases this rate to a point where gold ceases to dissolve. If silver is present in solution CaS will precipitate it and may prevent its final recovery.

Calcium sulphate, if present, crystallizes in pipe lines, launders, and tanks; it chokes filter and clarifier mats; it coats zinc-dust cloths and frames or settles out in zinc boxes, if the latter are in use; finally it coats zinc shavings or dust and thereby inhibits precipitation.

Due to the difficulties described, it is advisable to roast the ore so as to produce as small amounts of these two compounds as possible. It is difficult, however, to approach complete elimination of the sulphur in the case of the basic ores. Considerable quantities of $CaSO_4$ are usually present in the calcines but troublesome quantities of CaS may be largely avoided.

A study of the behavior of $CaSO_4$ at elevated temperatures with certain fluxes has been made by Hofman and Mostowitsch and most of the statements which follow on this subject have been taken from their report.[4]

When calcareous ores containing metallic sulphides are roasted the lime is converted almost quantitatively to $CaSO_4$, providing sufficient sulphur is present. When the $CaSO_4$, however, is heated in the presence of either CO or C, which happens in roasting operations, a reduction of the $CaSO_4$ takes place as indicated by the reactions which follow:

Reduction of $CaSO_4$ by CO: $CaSO_4 + 4\ CO = CaS + 4\ CO_2$.

This reaction is slow below 680°C., is rapid between 750° and 850°C. and is practically completed at 900°C. The reaction also take place without loss of sulphur.

4 Hofman, H. O., and Mostowitsch, W., The Behavior of Calcium Sulphate at Elevated Temperatures with Some Fluxes: Trans. Am. Inst. Min. Eng., vol. 39, 1909, p. 628.

Reduction of $CaSO_4$ with C: $CaSO_4 + 4\ C = CaS + 4\ CO.$
$CaSO_4 + 2\ C = CaS + 2\ CO_2.$

These reactions are fairly rapid at 700°C. and are completed at 1,000°C.

At the Golden Cycle plant CaS is formed as noted in these reactions. The carbon monoxide is introduced into the roaster with the fuel and the solid carbon from the cars which deliver ore to the plant, as many of these cars are used for the transportation of lignite coal to the mines. In addition to the solid carbon, introduced as noted, additional carbonaceous cinder from locomotives is introduced into the cars enroute to the plant.

This solid carbonaceous material if not completely burned during roasting of the ore forms a substance analogous to charcoal which readily precipitates gold in leaching operations. A roasting temperature high enough to completely burn this material also decomposes calcium and magnesium carbonates. The CO_2 gas liberated surrounds the ore particles for a time and prevents the oxidation of any CaS present to $CaSO_4$. Later during the roast the CaS is partly converted to $CaSO_4$ and CaO but some CaS remains in the final calcines. During leaching operations the small amount of CaS which goes into solution is soon oxidized or precipitated by zinc or iron salts. No satisfactory method has yet been found to prevent the formation of $CaSO_4$ during roasting or to remove it cheaply after it has deposited in various parts of the equipment.

The calcines drop from the cooling hearth of the roaster onto a reciprocating drag conveyor, 350 feet long. The hot ore is moved along the drag by sheet-iron fins 34 inches long by 8 inches deep. The conveyor requires 16 1/2 hp. for operation. The drag discharges the calcines to a conveyor belt which is specially prepared to handle hot material. The product when reaching this belt is at a temperature of about 85°C. and contains 2.5 percent of moisture; the water is added as a spray between the drag and belt conveyors.

Grinding

The belt conveyor delivers the calcine to five 6-foot Chilean mills for grinding. The Chilean mills are equipped with screens which have 0.0496-inch openings, and the grinding is done in cyanide solution.

Recovery of Coarse Gold

The pulp from the Chilean mills is fed to blanket tables 16 by 12 feet in size for the recovery of the so-called "coarse gold." Each table is divided into four sections with a 3-inch drop between sections. The blankets used are either ordinary cheap part wool, fairly lightweight cotton, or corduroy; they are changed twice in 24 hours. The concentrates caught by the tables are removed by stretching the blankets over a rail and washing with cyanide solution which is delivered by a pressure hose. The recovery of gold by these tables amounts to between 22 and 40 percent of the total gold.

Amalgamation

The concentrates washed from the blankets are directed into a small hopper-bottomed receiver and from there are fed to a small grinding pan. Mercury is added to the pan and the coarse free gold recovered by amalgamation. Since the amount of concentrates to be treated is small they are fed to the grinding pan at a slow rate and most of the minerals other than gold are ground fine enough by the mullers to overflow from the pan. The overflow pulp is added to the general mill circuit.

The amalgam is periodically drained through a plug located near the bottom of the pan and after cleaning is squeezed through a cloth which resembles bed ticking. The amalgam is retorted and the mercury returned to the mill for reuse. Fouling of mercury during pan amalgamation of concentrates is negligible; the average yearly loss of mercury for the past 3 years amounted to 275 pounds.

The retort bullion is melted and cast into bars for shipment to the mint at Denver. The gold bullion varies from 920 to 940 fineness in gold and from 45 to 55 fineness in silver.

Leaching of Sand

The tailings of the blanket tables and the overflow pulp of the grinding pan are delivered to a Dorr bowl classifier by centrifugal pumps. The classifier is equipped with a 12-foot-diameter bowl and is operated with liberal amounts of back-wash water added to the raking compartment; this results in the production of a sand practically free from slime. These sands amount to about 70 percent of the mill feed and contain from 22 to 26 percent of moisture. They are conveyed and distributed to the sand leaching vats by a rubber belt conveyor.

There are 11 leaching vats 50 feet in diameter and 7 feet deep; each will hold 600 tons of dry sand. The time of leaching varies from 6 to 8 days. The charges are drained and aerated four to five times during the leaching cycle for 8-hour periods.

The solution which overflows the tanks during charging and the solution drained during the first 48 hours of leaching flows into gold-solution storage tanks. From the storage tanks it is fed continuously to the precipitation presses.

After the 48-hour initial leaching period barren sump solution from the precipitation presses is used for leaching and washing purposes. These solutions upon leaving the leaching vats are either sent to the Chilean mills where the ore, as previously mentioned, is ground in cyanide solution or, if the solution is low grade, it may be pumped to one or two of the last tanks for leaching purposes. The leaching with barren sump solution is followed by a final water wash.

The leached sand charge is removed from vats by sluicing with water into launders which lead to a common sump. From the sump the tailings are pumped in three stages to the tailings dump. A charge of 600 tons is removed from a vat in about 3 1/2 hours by two men operating a 2 1/2-inch hose equipped with a 3/4-inch nozzle. Water at 200 pounds pressure is used.

The tabulation which follows presents a screen-assay analysis of the sand tailings from leaching operations.

Screen size, mesh	Weight, per cent	Gold assay, ounces per ton
Plus 20	18	0.01
Plus 30	28	.01
Plus 40	14	.01
Plus 60	25	.01
Plus 100	6	.02
Plus 150	5	.02
Plus 200	2	.03
Minus 200	2	.03
Composite	100	0.012

Leaching of Slime

The overflow pulp of the Dorr bowl classifier contains about 7 percent of solids. This pulp is thickened to 42 percent of solids in a 50-foot Dorr tray thickener, the overflow solution being returned to the Chilean grinding mills. The thickened pulp is discharged by four Dorrco diaphram pumps and is delivered to a 38- by 24-foot continuous Dorr agitator. The overflow from this agitator feeds another continuous Dorr agitator 34 by 14 feet in size. This agitator discharges into a Dorr bowl classifier, the sand returning to the first classifier and the overflow slime feeding a 30- by 9-foot continuous Dorr agitator which in turn overflows into a 30- by 9-foot continuous agitator of the paddle type. The pulp leaving here is rethickened to 50 percent of solids in two 30- by 9-foot Dorr tray thickeners. The overflow solutions from these thickeners go to a clarifier and from there to storage precipitation tanks.

Thickened pulp from the Dorr tray thickeners is pumped to a 37- by 23-foot continuous Dorr agitator which is followed by either of two 30- by 9-foot mechanical agitators. These latter agitators feed the Butters filters.

The total time of agitation amounts to between 65 and 75 hours. The solids handled contain 4 to 5 percent 100-mesh laboratory screen oversize and about 84 percent minus 200-mesh material.

Filtering

The filter equipment comprises two Butters intermittent type of filters each equipped with 87 canvas-covered 5- by 9 1/2-foot frames. The filters are fed from the agitators as noted, the agitators being filled alternately.

The slime cake is given a barren-solution wash which is followed by a short water wash. The tailings from the filter presses contain 0.0276 ounce of gold per ton of which $0.0245 is soluble gold. The filtered solutions pass to a clarifier and from there to the precipitation presses.

Clarification of Solutions

Clarification at this plant is best effected by means of the Hardinge sand clarifier; this applies to solutions from the slime plant before precipitation. The clarifier used is 30 feet in diameter and has a capacity of 3,000 tons of solution per day. It is equipped with a sand filter bed which is kept clean by a spiral scraper that discharges the sludge in a manner similar to that of an ordinary thickener.

Precipitation

Zinc shavings contained in zinc boxes were formerly used for the precipitation of gold from cyanide solutions. Since 1929, precipitation has been effected with zinc-dust and Merrill presses have been used for the recovery of the precipitate. The solutions are passed through Crowe vacuum equipment before precipitation. The barren solution, which contains 0.02 ounce of gold per ton, flows into sumps and is used as wash solution for both sand and slime treatments.

At times it has been found difficult to satisfactorily precipitate the values from the cyanide solutions derived from the treatment of roasted Cripple Creek ores. The factors which follow have been found to influence the results of precipitation: (a) Type of ore, whether basic or otherwise; (b) quantities of sulphates and sulphides formed during roasting operations; (c) rate of feed to roasters, and (d) temperature used in roasting.

The addition of lead acetate, in amounts depending upon conditions in roasting, has been found at times to be beneficial.

Treatment of Precipitates

When zinc shavings were used for precipitation the zinc boxes were cleaned up by removing the contents from the first or first and second compartments of the seven compartment boxes. The precipitate was treated with sulphuric acid in a tank. When most of the zinc had dissolved, the sludge was dropped into a small air-tight tank and from there forced by air pressure into a small clean-up press. The gold sludge was washed in this press with hot water for the removal of zinc sulphate. The press was then dismantled,

the sludge cake removed, and charged into a muffle type furnace. The muffle was brought to a red heat and most of the lead and any remaining zinc oxidized. The roasted product was mixed with flux in the proportion of 100 pounds of residue to 84 pounds of flux. The flux contained 28 pounds of silicious sand, 28 pounds of bicarbonate of soda, 14 pounds of borax glass, and 14 pounds of fluorspar. The charge, contained in graphite crucibles, was melted in oil-fired tilting furnaces.

When the change from zinc shavings to zinc-dust was made the sulphuric acid treatment was discontinued. The precipitate, after removal from the Merrill press, is now put into iron pans and mixed, while wet, with sodium nitrate in the proportion of 125 pounds of precipitate to 25 pounds of nitrate. The pans are placed in a furnace equipped with cast-iron muffles and the charge is heated to a red heat. The heating results in the drying and sintering of the charge which may be subsequently handled without loss of dust. The bullion is then mixed with the sand-borax glass-soda nitrate-fluorspar flux, previously noted, and charged to a fire-clay tile lined Monarch-Rockwell type of furnace for melting.

METALLURGICAL DATA AND RECOVERY

Screen analyses of various plant intermediate and final products for the year 1929 are given in table 1. Metallurgical data for the year 1929 are presented in table 2. A summary of metallurgical results for 1929 which includes distribution of products, assays, and recoveries is given in table 3.

The recovery of gold at this plant has been found to be affected by the conditions which follow:

(1) Very careful attention to the details of the roasting of basic ores is necessary.

(2) The complete removal of free gold particles of any appreciable size by blankets, or some other means, is essential if tailings of low gold content are to be obtained within reasonable time periods.

(3) The consumption of chemicals, the formation of calcium sulphate and other cementing compounds and the amounts of deleterious sulphides produced vary with the types of ores and with the manner of roasting.

(4) Recovery of gold has at times decreased even when the roast appears satisfactory, and this condition is apparently due to the accumulation of zinc salts in the cyanide solutions. Sufficient amounts of sulphides are usually present in the calcines to precipitate injurious soluble zinc; but when the recovery decreases, the addition of soluble sulphides is in general beneficial.

(5) As a general condition, both sand and slimes are difficult to wash. At the beginning of washing operations satisfactory displacement of gold-bearing solution apparently takes place but later in the washing operation the dissolved gold appears to be removed chiefly by diffusion rather than displacement.

It has been found that the use of what might be termed "excessive barren washes" for the removal of dissolved values in both sands and slimes is advantageous.

(6) In a plant of this character mechanical leaks of ore or solution and waste of values of any kind should be very carefully watched.

COSTS

A summary of plant costs for 1929 is presented in table 4. The distributions of labor and power are given in tables 5 and 6, respectively.

Table 1. - Screen analyses of plant products for 1929

Screen size	Weights, percent									
	Feed to 40-inch rolls	Feed to Symons cone crusher	Feed to comminuters	Feed to 6-mesh Hum-mer screens	Roaster feeds		Feed to Chilean mills	Dorr bowl classifier		
					Coarse	Fine		Feed	Sand	Slime
Plus 3-inch, square	17.2	---	---	---	---	---	---	---	---	---
Plus 2-inch, square	15.5	7.7	---	---	---	---	---	---	---	---
Plus 1-inch, square	28.0	42.2	---	---	---	---	---	---	---	---
Plus 13/16 inch, square.	4.7	13.2	---	---	---	---	---	---	---	---
Plus 9/16 inch, square..	5.4	15.7	---	---	---	---	---	---	---	---
Plus 3/8 inch, square...	6.6	13.2	6.2	---	---	---	---	---	---	---
Plus 5/16 inch, square .	3.0	3.1	11.8	---	---	---	---	---	---	---
Plus 8 mesh	7.8	2.8	76.1	14.7	95.7	4.0	14.7	---	---	---
Plus 10 mesh	2.1	.2	1.9	16.0	4.3	17.5	16.0	---	---	---
Plus 14 mesh	.9	.1	.8	10.6	---	12.0	10.6	---	---	---
Plus 16 mesh	.4	.2	.2	2.1	---	2.4	2.1	---	---	---
Plus 20 mesh	1.0	.2	.3	7.6	---	8.6	7.6	12.6	18.0	---
Plus 30 mesh	1.6	.2	.2	11.0	---	12.5	11.0	19.6	28.0	---
Plus 40 mesh	.7	.1	.1	7.5	---	8.5	7.5	9.8	14.0	---
Plus 60 mesh	2.4	.6	1.4	11.0	---	12.4	11.0	17.5	25.0	---
Plus 80 mesh	.1	.1	---	.7	---	.8	.7	---	---	0.1
Plus 100 mesh	.7	.1	.1	2.5	---	2.8	2.5	5.5	6.0	4.1
Plus 150 mesh	.6	.1	.1	3.1	---	3.5	3.1	4.9	5.0	4.8
Plus 200 mesh	.7	.1	.3	2.8	---	3.2	2.8	3.5	2.0	7.0
Minus 200 mesh	.6	.1	.5	10.4	---	11.8	10.4	26.6	2.0	84.0
Totals	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Table 2. - Metallurgical data for 1929

Total ore treated .... tons	298,500
Days operated	365
Operating time per day .... hours	24
Operating time .... percent	100
Average ore treated per 24 hours .... tons	817.8
Assay of plant heads .... ounces gold per ton	.502
Recoveries of gold:	
By amalgamation .... percent	25.33
By cyanidation before classification do.	55.10
Cyanide sand leaching .... do.	10.70
Cyanide slime leaching .... do.	5.68
Combined recovery .... do.	96.81
Losses of soluble gold in tailings:	
Sand tailings .... per ton	$0.0132
Slime tailings .... do.	$ .0245
Average .... do.	$ .0166
Consumption of reagents, pounds per ton of ore treated:	
Cyanide, 100 per cent sodium cyanide	.964
Hydrated lime	6.119
Zinc-dust	.269
Hydrochloric acid	.169
Lead acetate	.037
Strength of solutions, pounds sodium cyanide per ton of solution	2.0 to 0.7
Consumption of balls - pounds per ton of ore treated.....	.378
Consumption of liners .... do.	.105
Consumption of Chilean mill tires and dies do.	.053
Net water consumption, including power plant, gallons per ton of ore treated	1,045

Table 3. - Summary of metallurgical results for 1929

	Moisture, percent	Dry weight, tons	Dry weight, percent of total	Assay, ounces gold per ton	Weight of gold, ounces	Percent of total gold	Percent of total gold	
							Recovered	Lost
Plant heads	3.7	298,500	100.00	0.502	149,847	100.00	---	--
Heads to roasters ..	3.6	298,500	100.00	.502	149,847	100.00	---	--
Roaster calcines ...	---	285,366	95.60	.525	149,817	99.98	---	0.02
Amalgamation	---	---	---	---	37,956	25.33	25.33	--
Cyanidation during amalgamation	---	---	---	---	82,571	55.10	---	--
Dorr bowl classifier heads	---	285,366	95.60	.1026	29,290	19.55	---	--
Heads to sand leaching	26.2	199,756	66.92	.0922	18,418	12.29	---	--
Solutions from sand leaching	---	---	---	---	16,041	10.70	---	--
Tailings from sand leaching	11.2	199,752	66.92	.0119	2,377	1.59	---	--
Heads to slime leaching	93.1	85,610	28.68	.1270	10,872	7.26	---	--
Solutions from slime leaching	---	---	---	---	8,509	5.68	---	--
Tailings from slime leaching	32.8	85,608	28.68	.0276	2,363	1.58	---	--
Combined plant tailings	17.7	285,360	95.60	.0166	4,737	3.17	---	3.17
Combined cyanide solutions	---	---	---	---	107,121	71.48	71.48	--
Totals	---	---	---	---	---	---	96.81	3.19

Table 4. - Summary of plant costs for 1929

	Operating labor	Supervision	Power [1]	Supplies	Reagents	Repairs Labor	Repairs Other	Miscellaneous	Total
Unloading, operations and repairs combined...	$0.0680	---	---	$0.0030	---	---	---	---	$0.0710
Sampling, including jaw crushers and rolls...	.0576	---	$0.0099	.0181	---	$0.0114	$0.0146	$0.0274	.1390
Symons crusher	.0095	---	.0112	.0364	---	.0059	.0004	---	.0634
Comminuters	.0143	---	.0379	.0268	---	.0161	.0206	---	.1157
Screens and elevators	.0079	---	.0049	.0047	---	.0030	.0001	---	.0206
Roasters	.1333	---	.0464	.3501	---	.0172	.0325	.0556	.6351
Chilean mills	.0161	---	.0644	.0259	---	.0141	.0044	---	.1249
Conveying	.0520	---	.0321	.0194	---	.0191	.0087	---	.1313
Amalgamation, operations and repairs combined	.0150	---	.0013	.0060	---	---	---	---	.0223
Classification	.0035	---	.0010	.0011	---	.0017	.0010	---	.0083
Sand leaching	.0287	---	.0035	.0072	---	.0042	.0039	---	.0475
Slime leaching	.0142	---	.0156	.0061	---	.0035	.0035	.0156	.0585
Filters	.0152	---	.0052	.0120	---	.0039	.0028	---	.0391
Clarifier	.0061	---	.0013	.0033	---	.0002	.0001	---	.0110
Precipitation	.0129	---	.0010	.0204	$0.0331	.0017	.0007	---	.0698
Refining	.0081	---	.0026	.0265	---	.0013	.0009	.0025	.0419
Disposal of tailings	.0237	---	.0168	.0176	---	.0057	.0045	---	.0683
Assaying	.0245	---	---	.0234	---	---	---	---	.0479
Reagents:									
Cyanide	---	---	---	---	.1543	---	---	---	.1543
Lime	---	---	---	---	.0394	---	---	---	.0394
Lead acetate	---	---	---	---	.0051	---	---	---	.0051
Hydrochloric acid	---	---	---	.0035	---	---	---	---	.0035
General supervision and superintendence	---	$0.0392	---	---	---	---	---	---	.0392
Office	.0296	---	---	.0120	---	---	---	---	.0416
Yarding	---	---	---	---	---	---	---	.0175	.0175
Bullion	---	---	---	---	---	---	---	.0355	.0355
Outside lighting	---	---	---	---	---	---	---	.0016	.0016
By-product	---	---	---	---	---	---	---	.0113	.0113
Flue-dust	---	---	---	---	---	---	---	.0107	.0107
Umpire expense	---	---	---	---	---	---	---	.0020	.0020
Miscellaneous buildings	---	---	---	---	---	---	---	.0139	.0139
Railroad tracks	---	---	---	---	---	---	---	.0071	.0071
Water works line	---	---	---	---	---	---	---	[2] .0118	.0118
Track scales	---	---	---	---	---	---	---	.0003	.0003
Watchmen and fire protection	.0052	---	---	---	---	---	---	.0062	.0114
All insurance	---	---	---	---	---	---	---	.0336	.0336
General expense	---	---	---	---	---	---	---	.0585	.0585
Totals	0.5454	0.0392	0.2551	0.6235	0.2319	0.1090	0.0987	0.3111	2.2139

1/ Power costs $0.0095 per kilowatt-hour.
2/ Water costs $0.01096 per 1,000 gallons.

Table 5. - Distribution of labor for 1929

	Tons of ore treated per 8-hour man-shift			Percent of total labor
	Operation	Maintenance	Total	
Unloading	60.904	(1)	60.904	8.463
Sampling	81.008	359.584	66.111	7.796
Symons crushers	342.288	1,859.456	289.076	1.783
Comminuters	323.005	704.440	221.456	2.328
Screens and elevators .	393.144	860.000	269.804	1.911
Roasters	26.126	247.776	23.632	21.816
Chilean mills	242.256	241.400	120.912	4.262
Conveying	72.067	130.136	46.382	11.113
Amalgamation	240.279	(1)	240.279	2.145
Classification	745.128	(1)	745.128	.692
Leaching	220.749	745.416	170.296	3.025
Slimes	257.357	657.328	184.944	2.787
Filters	307.144	1,349.016	250.184	2.064
Clarifier	845.904	(1)	845.904	.609
Precipitation	328.140	(1)	328.140	1.571
Refining	461.744	(1)	461.744	1.116
Tailings	172.720	742.448	140.120	3.679
Assaying	277.416	(1)	277.416	1.858
General supervision and superintendence	273.376	(1)	273.376	1.886
Power plant	135.522	430.000	103.048	5.001
General expense	37.283	(1)	37.283	14.095

1/ Maintenance included in operation.

Table 6. - Distribution of power for 1929

	Kilowatt-hours per ton of ore treated	Per cent of total power
Sampling, including jaw crushers and rolls	1.2575	4.56
Symons crusher and comminuters	5.1328	18.59
Roasting	4.9574	17.96
Chilean mills	6.7256	24.37
Classifiers, screens, elevators, and conveyors	3.9562	14.33
Amalgamation	.1302	.47
Cyanidation:		
Sand leaching	.3856	1.40
Slime leaching:		
Agitating and settling	1.6241	5.88
Filtering	.5414	1.96
Aeration	.6416	2.32
Precipitation	.0987	.36
Refining	.1442	.52
Disposal of tailings	1.7490	6.34
By-product	.2594	.94
Totals	27.6037	100.00

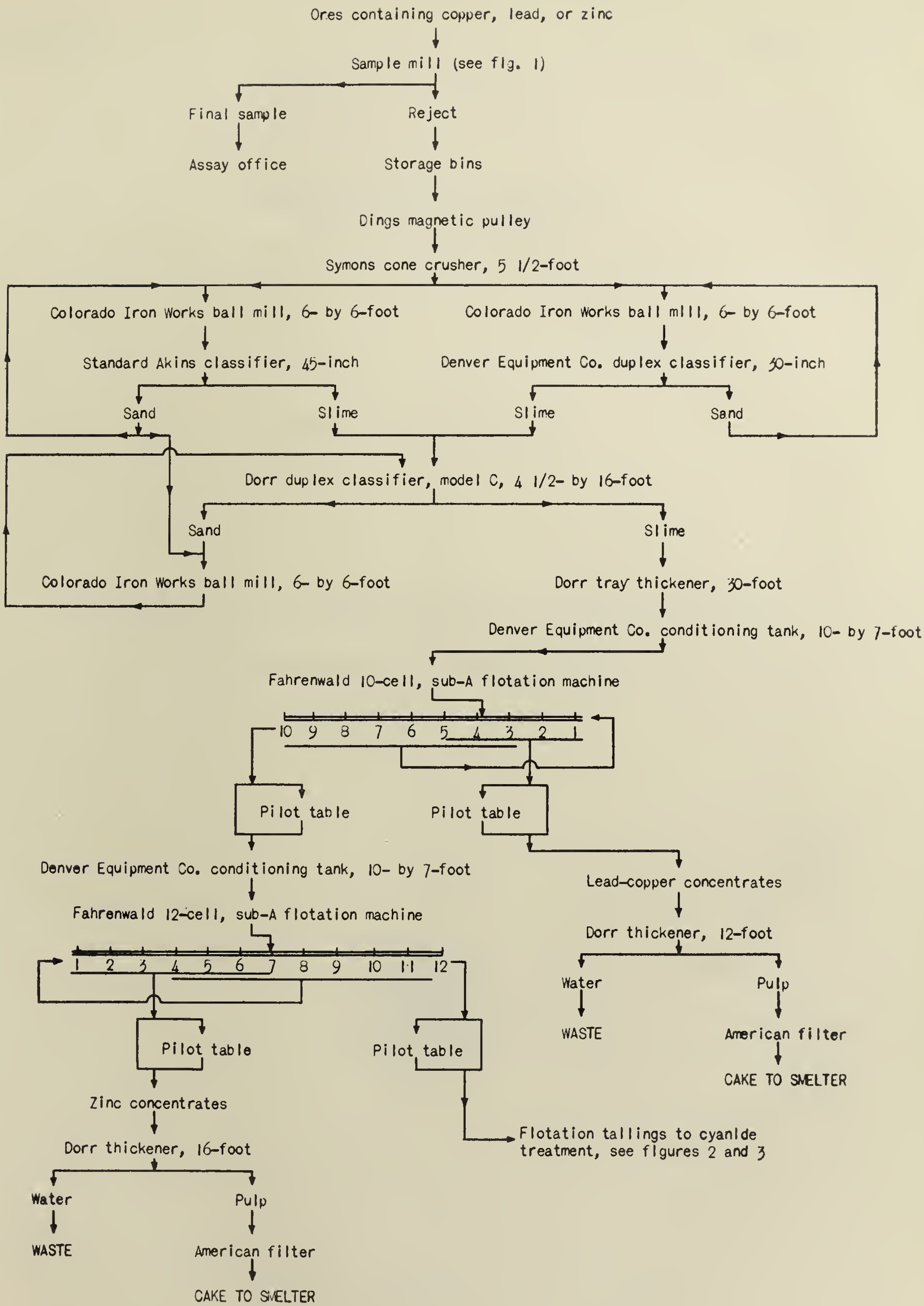

Figure 4.- Flow sheet of treatment for ores containing lead, copper, or zinc.

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INFORMATION CIRCULAR

MILLING METHODS AND COSTS AT THE CONCENTRATOR OF THE PREMIER GOLD MINING CO., LTD., PREMIER, B. C., CANADA

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MILLING METHODS AND COSTS AT THE CONCENTRATOR OF THE PREMIER GOLD MINING CO. LTD., PREMIER, B. C., CANADA[1]

By D. L. Pitt,[2] W. J. Asselstine,[3] and D. L. Coulter[4]

INTRODUCTION

This paper, describing the milling methods at the concentrator of the Premier Gold Mining Co., Ltd., Premier, B. C., Canada, is one of a series being prepared by the United States Bureau of Mines on milling methods and costs in the various mining districts throughout the United States and Canada.

ACKNOWLEDGMENTS

The authors wish to acknowledge the assistance and suggestions given during the preparation of this paper by the engineering staff and accounting department of the Premier Gold Mining Co., Ltd., all of whom have assisted materially in gathering the data used.

LOCATION

The Premier mine and mill are situated in northern British Columbia, approximately 15 miles north of the head of the Portland Canal in the Salmon River section, about half a mile east of the Alaskan-British Columbia boundary line where it crosses the Salmon River. The mill is located on a rather steep hillside on the horizon of the main haulage level of the mine, and is within 100 feet of the mouth of the tunnel.

Both the mine and mill are connected with tidewater at Stewart, B. C., by an aerial tramway 11.5 miles long, which delivers both ore and concentrate to the bunkers at the dock for shipment to the smelters. All the mine and mill supplies except heavy machinery, are transported from Stewart dock to

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6742."

2 One of the consulting engineers, U. S. Bureau of Mines, and manager, Premier Gold Minind Co., Ltd.

3 Mill superintendent, Premier Gold Mining Co., Ltd.

4 Assistant mill superintendent, Premier Gold Mining Co., Ltd.

Premier over this tramway. Heavy pieces are hauled either by truck in summer or sleighs in winter over a good road which runs from Stewart dock to Premier.

HISTORY

When the Premier mine first came to the attention of the public, in 1918, the impression was general that it was a bonanza deposit of straight shipping ore. While the first shipments were very high-grade gold-silver ore, it was early recognized by the management that in addition to the high grade there would be a considerable tonnage of milling ore. It was therefore necessary to look toward a milling program early in the mine's history.

The ore developed during the early stages of the mine's operation was a highly siliceous deposit carrying an intimate mixture of auriferous pyrite with small percentages of zinc, lead, and copper sulphides. The predominating values were in silver and gold.

Samples taken from development work and diamond-drill cores were sent to the Flat River, Mo., testing plant of the American Smelting & Refining Co., where tests were run to determine the best method of treating this type of ore. The wide variation in the composition and value of the ore is illustrated by the following analyses, representing various samples on which tests were run.

Table 1.- Variations in analyses of Premier ores

Assay No.	Assay, ounces per ton		Assay, percent					
	Gold	Silver	Lead	Zinc	Copper	Iron	Insoluble	Sulphur
1	0.024	14.2	0.20	....	0.015	3.6	91.1	1.65
2	.40	14.9	.15	....	.005	4.8	88.1	2.90
3	.41	5.2	2.45	....	.28	15.2	57.6	18.05
4	.64	20.6	.32	....	.025	6.8	80.7	2.85
5	1.26	28.6	.65	1.2	.....	6.3	80.3	6.50
6	1.33	42.9	.78	1.1	.....	6.1	77.3	6.30

The treatment methods used in testing comprised straight cyanidation and cyanidation in combination with tabling and flotation. The method adopted as a result of a great many tests, was to remove a fairly coarse table concentrate, high in gold; the table tailing was reground and treated by flotation for the removal of additional gold and silver, and the flotation tailing was thickened and treated by cyanide, using countercurrent decantation. This method differed from any other commercial installation then in use, and was distinctive in the feature that all grinding and flotation operations were done in cyanide solution. Flotation reagents used comprised: steam-distilled pine oil, Barrett No. 634 creosote oil, and coal tar. Wilfley tables and K&K flotation machines were used in concentrating. This treatment method was employed until 1926, with the results indicated in tables 2A and 2B.

Table 2A.- Weights and assays of mill products, 1923 to 1926

Year	Mill heads			Table concentrates			Flotation concentrates			Cyanide precipitates		
	Weight, dry tons	Assays, ounces per ton		Weight, dry tons	Assays, ounces per ton		Weight, dry tons	Assays, ounces per ton		Weight, dry tons	Assays, ounces per ton	
		Gold	Silver		Gold	Silver		Gold	Silver		Gold	Silver
1923	57,796	0.55	9.90	5,713	3.64	20.07	1,731	0.71	131.30	11.5795	726.66	9.118.7
1924	61,966	.54	10.40	6,427	3.30	23.59	2,148	1.48	111.26	13.642	510.80	8,674.7
1925	55,705	.47	8.30	2,639	6.27	26.19	3,718	1.65	82.44	6.4925	427.61	4,038.7
1926	122,152	.41	9.08	3,293	6.65	69.55	16,704	1.47	46.11	....	......	....

Table 2B.- Recoveries and concentration ratios, 1923 to 1926

Year	Recoveries, percent								Concentration ratios, percent of heads	
	Tables		Flotation		Cyanidation		Total			
	Gold	Silver	Gold	Silver	Gold	Silver	Gold	Silver	Tables	Flotation
1923	65.42	20.04	3.86	39.73	26.46	18.44	95.74	78.21	9.88	3.00
1924	63.38	23.53	9.50	37.08	20.83	18.36	93.71	78.97	10.37	3.47
1925 1/	63.20	14.95	23.43	66.29	10.60	5.67	97.23	86.91	4.74	6.67
1926 2/	43.29	20.66	48.61	69.11	.....	.....	91.90	89.77	2.69	13.68
Average	.....	......	.....	......	.....	.....	94.48	84.55	......	....

1/ Cyanided for five months only.
2/ New 250-ton unit started in March.

In 1926 the development and use of chemical flotation reagents was receiving considerable attention and experiments with various combinations of these reagents were carried out in the mill laboratory. It was found that when xanthate and steam-distilled pine oil were used in a circuit made slightly alkaline with sodium carbonate, the recoveries by tabling and flotation equaled those formerly obtained by tabling, flotation, and cyanidation and at a considerable saving in cost. The results were so gratifying that a mill run of one month's duration was made using the experimental method and reagents as described. The results of the mill run were entirely satisfactory, and it was decided to discontinue cyanidation.

The original mill had a maximum daily capacity of 150 tons. When cyanidation was discontinued, another unit of 250 tons capacity was built. This increase of capacity was justified by additional milling ore that had been developed while opening up the mine.

The treatment methods employed for approximately one year after the completion of the new unit continued to be tabling followed by flotation. During this period experiments were carried on in the mill laboratory using various other combinations of chemicals and reagents. The results of these experiments indicated that all-flotation methods using aerofloat and sodium carbonate with a small amount of cyanide would give recoveries equally as good as were being made with flotation and tables and, therefore, in 1927, the tables were eliminated in the flow sheet and all-flotation methods adopted.

The kinds and quantities of flotation reagents used after the adoption of all-flotation methods from 1927 to 1931, inclusive, are given in table 3, and the metallurgical data for the same periods are presented in table 4.

In table 3 a decided decrease in the amount of sodium carbonate used may be noted, beginning in 1929, which was due to the adoption of hydrogen-ion control in the flotation circuit. The highest recoveries have been made when a pH value of 7.8 was maintained, although pH values greater than 7.8 have had neither detrimental nor beneficial results.

PRESENT METHOD OF MILLING

The ore is hauled to the mill by Edison storage-battery locomotives in trains of nine 2-ton-capacity cars. The main haulage system is on the fourth level of the mine and at the same elevation as the top of the mill bins. The ore was weighed in cars over a period of years and a weight factor determined for the cars. At present the mill tonnage is determined from the number of cars delivered, the weight factor for the cars being checked periodically. Moisture samples are taken, and over a period of years the ore has averaged 2 percent of moisture.

A side elevation of the concentrator is shown in figure 1.

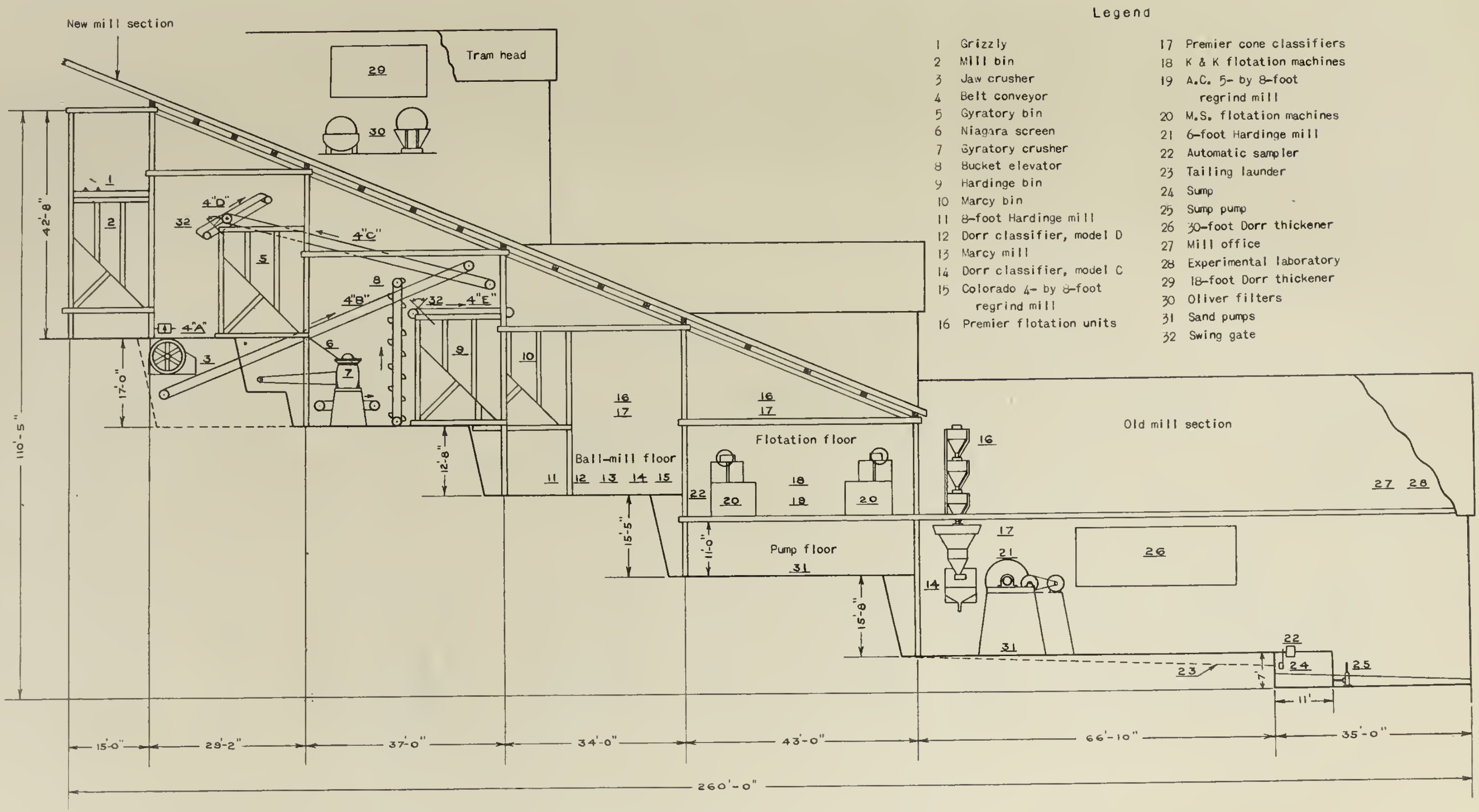


Figure 1.- Elevation with transverse section of Premier concentrator.

Table 3.- Reagents used in all-flotation treatment

Reagent, pounds per ton milled	1927	1928	1929	1930	1931
No. 1 pine oil	0.13	-	-	-	-
Potassium xanthate	.06	-	-	-	-
Sodium carbonate	.52	0.40	0.08	0.02	0.03
No. 634 creosote	.13	-	-	-	-
Sodium cyanide	.07	.09	.07	.03	.01
Aerofloat	.26	.27	.27	.28	.27

Table 4.- Metallurgical data for years 1927 to 1931, inclusive

Year	Mill heads			Concentrates							Recoveries, percent		Concentration ratios, percent
	Weight, dry tons	Assays, ounces per ton		Weight, dry tons	Assays, ounces per ton		Analyses, percent						
		Gold	Silver		Gold	Silver	Lead	Zinc	Iron	Insoluble	Gold	Silver	
1927	147,536	0.39	7.17	1/1,701	5.27	21.89	3.7	5.4	37.0	6.6	15.41	3.51	1.15
				23,995	1.95	42.58	6.3	10.5	28.6	12.6	80.24	85.43	16.25
1928	162,112	.38	6.73	17,062	3.47	53.46	8.1	12.1	21.2	20.7	95.30	84.04	10.52
1929	165,143	.30	6.38	14,631	3.11	61.24	6.1	11.0	19.4	26.9	92.65	80.77	8.86
1930	151,936	.30	6.95	15,681	2.71	62.19	4.3	8.1	23.7	26.8	93.37	92.34	10.32
1931	169,760	.305	5.93	20,116	2.41	43.17	3.3	8.7	26.0	20.7	93.35	86.29	11.85

1/ Table concentrate produced during first 6 months.

CRUSHING

Referring to figure 2, mine-run of ore is dumped onto a grizzly made of 80-pound rails spaced 12 inches apart and thence into two bins, each of 100-ton capacity, one for high-grade ore and the other for mill ore. The high-grade ore is drawn from its bin onto a 26-inch, 12-ply belt conveyor, which carries it to an 18- by 30-inch Blake-type crusher set at 3 inches. The crushed product is then elevated by a system of three conveyors to the tramway ore bin, from which it is drawn into tram buckets and sent to the bunkers at the dock for shipment to smelters at either Tacoma or Anyox.

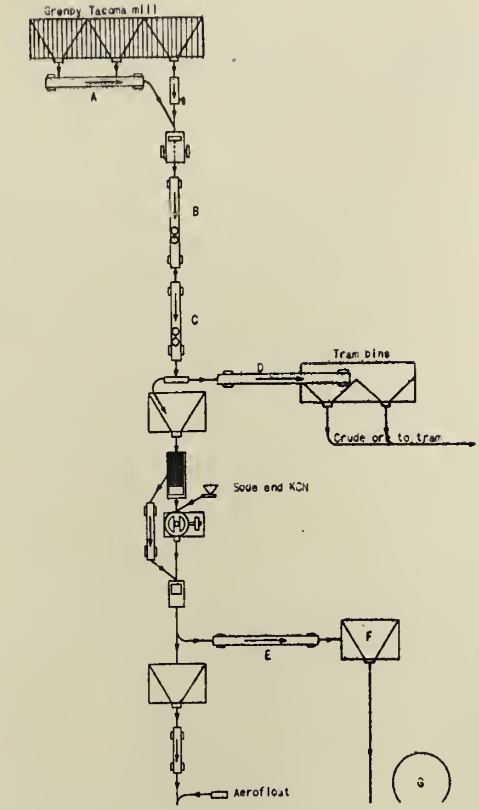
The mill ore is drawn from its bin onto a 42-inch Stephens-Adamson apron feeder and is fed directly to the same 18- by 30-inch jaw crusher that handles smelting ore. From the jaw crusher the ore is elevated by a series of two inclined conveyors, and discharged into a 100-ton-capacity bin. An electromagnet is suspended over each conveyor for the removal of tramp steel and iron.

From the bin the Blake crusher product passes over a Niagara screen with 1-inch openings. The oversize is fed directly to a No. 6 McCully Superior gyratory crusher set at 1 inch. The material passing through the screen unites with the crusher discharge, then elevated by a 14- by 7-inch bucket elevator, and is discharged against an adjustable gate so set that about two-thirds of the product from the elevator goes into a bin above the Hardinge mill while the remainder is carried by a 20-inch conveyor to a bin above the Marcy mill.

In the original crushing installation there were two primary No. 3D Gates gyratory crushers, one for mill ore and the other to handle shipping ore. It was soon found, however, that the run-of-mine ore contained considerable amounts of material too coarse to be handled by these small crushers; hence, it was necessary to put in the 18 by 30 inch jaw crusher as best the contour of the hill and the mill location would permit. The No. 3D mill gyratory crusher remained in the circuit until the new mill unit was built in 1926, when a larger one—a No. 6 McCully—replaced it to serve both old and new mill units. To accommodate these changes, a series of belt conveyors had to be worked out and fitted in as best suited to the existing units.

The capacity of the two crushers used at present is influenced by the character and condition of the ore mined during different seasons of the year. The heavy run-off when snows melt in the spring and during the rainy season finds its way underground and at times makes the ore very wet and sticky, causing the plugging of the gyratory crusher in spite of the removal of fines before crushing, thereby limiting the amount of material crushed. During these seasons the discharge opening of the gyratory crusher is increased somewhat, resulting in a coarser feed to the grinding units.

Crushing-plant data are given in table 5 and screen analyses of crushing-plant products in table 6.



Legend

Three mine bins, 100-ton capacity

Coarse crushing

A — Belt conveyor, 26-inch, 10-hp. motor

Apron feeder, 42-inch, 5-hp. motor

Blake crusher, 30- by 18-inch, set at 3 inches, 50-hp. motor

B — Belt conveyor, 22-inch, 350 feet per minute, 10-hp. motor

Suspended magnet

C — Belt conveyor, 22-inch, 350 feet per minute, 10-hp. motor

Suspended magnet

Deflecting gate - feed to tram or mill bins

D — Belt conveyor, 20-inch, 350 feet per minute to tram bins, 10-hp. motor

Fine crushing to 1 inch

Mill bin, 100-ton capacity

No. 2 Niagara screen, 1-inch openings

Dry reagent feeder for soda ash and cyanide

No. 6 McCully gyratory crusher, 50-hp. motor

Conveyor to elevator

Bucket elevator, 14- by 7-inch buckets, 10-hp. motor

E — Belt conveyor, 20-inch, 250 feet per minute, [illegible]-hp. motor

F — Bin feeding Marcy mill, 150-ton capacity

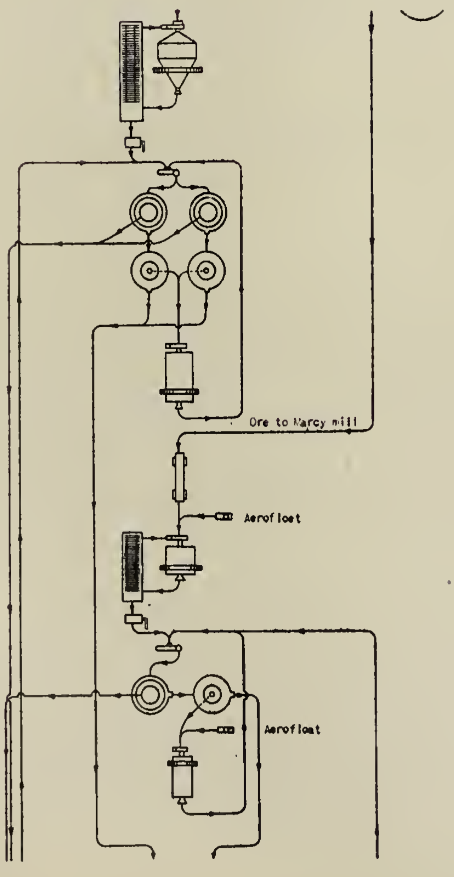
Bin feeding Hardinge mill, 100-ton capacity

Belt feeder, 20-inch, 6 feet per minute, 5-hp. motor

G — Water-supply tank, constant head

Mechanical reagent feeder

Figure 2.- Flow sheet of coarse and fine crushing.



Hardinge unit

Hardinge ball mill, 8-foot by 30-inch 25 r.p.m., 150-hp. motor, Tex-rope drive

Dorr classifier, model D, 6- by 20-foot

Galigher automatic sampler

Wilfley sand pump, 4-inch, 20-hp. motor

Two Premier flotation cells

Two Premier classifiers, 4-foot

A.C. tube mill, 5- by 8-foot, 28 r.p.m., 75-hp. motor

Belt feeder, [illegible] feet per minute

Marcy unit

Mechanical reagent feeder

No. 64 1/2 Marcy ball mill, 25 r.p.m.[1]

Dorr model C duplex classifier, 4 1/2-foot

Galigher automatic sampler

Wilfley sand pump, 4-inch, 20-hp. motor

Premier flotation cell

Premier classifier, 4-foot

Mechanical reagent feeder

Colorado Iron Works tube mill, 4- by 8-foot, 30 r.p.m., belt driven

[1] A 200-hp. motor drives entire Marcy unit.

Figure 3.- Flow sheet of grinding and preliminary flotation.

Table 5.- Crushing data for the year 1931

	Jaw crusher	Gyratory crusher
Size of feed openings..............inches ..	18 by 30	6
Size of discharge openings......... do. ...	3	1 1/4
Horsepower installed..........................	50	60
Type of drive.....................................	Belt	Belt
Wear of stationary plates.......tons crushed	22,266	---
Wear of swing plates............... do. ...	39,516	---
Wear of set of concaves............ do. ...	---	39,868
Wear of mantle...................... do. ...	---	106,386
Kind of wearing parts..........................	Manganese steel	Manganese steel
Ore crushed per hour............... tons ...	42	22

Table 6.- Screen analyses of crushing-plant products, 1931

Size	Blake crusher1/		Gyratory crusher		Niagara screen undersize		Ball-mill feed	
	Per-cent	Cumula-tive	Per-cent	Cumula-tive	Per-cent	Cumula-tive	Per-cent	Cumula-tive
Plus 3 inch...	34.5	34.5	41.2	41.2	....	....	....	....
Plus 1 inch...	39.5	74.0	56.0	97.2	....	....	18.2	18.2
Plus 1/2 inch.	15.2	89.2	2/2.8	....	44.6	44.6	33.1	51.3
Plus 1/4 inch.	4.3	93.5	....	....	20.1	64.7	17.6	68.9
Minus 1/4 inch	6.5	....	....	....	35.3	....	31.1	....

1/ Feed all minus 12 inch.
2/ Minus 1 inch.

GRINDING

The flow sheets of the newer and older grinding sections are presented in figure 3. In the newer unit the crushing-plant product is fed by a 20-inch conveyor to an 8-foot by 36-inch Hardinge ball mill operating in closed circuit with a Model D, 6- by 20-foot Dorr classifier. The overflow of the classifier is elevated by a 4-inch Wilfley sand pump to two 4 1/2-foot cones with 75° side slope placed in parallel and equipped with Premier flotation cells and sand discharge boxes. The two cones, with attachments, are in closed circuit with a 5- by 8-foot Allis-Chalmers tube mill.

In the older section of the mill, as shown in figure 3, the ore is fed by a 20-inch conveyor to a No. 64 1/2 Marcy ball mill operating in closed circuit with a 4-foot 6-inch by 14-foot 8-inch Model C Dorr classifier. The grate discharge of the ball mill has 1/8-inch openings. The classifier overflow is pumped to one 4 1/2-foot cone equipped with a Premier flotation cell and a sand-discharge box. The cone is operated in closed circuit with one 4- by 8-foot regrind mill.

The function of the flotation cells in both grinding circuits is to remove a coarse concentrate as soon as possible. The concentrates so removed from both circuits are sent by gravity to one 12-foot K&K cleaner flotation cell which produces finished concentrate for dewatering.

Data for primary and regrinding mills are presented in table 7; a typical screen analysis of feed to primary grinding mills is given in table 6 and screen analyses of primary grinding circuit products are presented in table 8; screen analyses of regrinding circuit products are presented in table 9. Consumption and costs of balls and liners are given in tables 10 and 11, respectively.

FLOTATION

The flotation flow sheet, except that part already described in the grinding circuits, is given in figure 5. The overflow from the cones of each grinding circuit passes to a set of three Premier flotation cells arranged in series. The concentrates from the six cells of the two sections are cleaned in one 12-foot K&K cell, which produces finished concentrate for dewatering. The tailing from the final cell of each section is sent to one 8-foot cone classifier. The discharge products of the two cone classifiers in both sections are sent to one 4 1/2-foot Dorr classifier. The Dorr classifier sand is delivered to one 6-foot by 16-inch Hardinge ball mill. The Dorr classifier overflow, together with the ball-mill discharge, is returned to the cone classifiers, thus closing the circuit between the cone classifiers and the grinding mill.

The overflows of the 8-foot cones are conveyed to a 4-foot surge box by a 4-inch Wilfley pump and from there are distributed partly to two 12-cell, 24-inch Minerals Separation subaeration machines placed in series, the remainder going to two double spitz, 12-foot K&K flotation machines placed in series. The concentrates from the first four cells of No. 1 Minerals Separation machines are finished products and are delivered to the concentrate collecting thickener by a 2-inch Wilfley pump. The froth products of the remaining 8 cells of this machine join those from the 12 cells of the second M. S. machine and are returned as middlings by a 2-inch Wilfley pump to both of the 4 1/2-foot cones above the regrind mills.

The concentrate from the first K&K rougher is cleaned in the K&K cleaner cell, which handles the rougher concentrates from the Premier cells. The froth from the second K&K rougher is returned to the 4 1/2-foot cones of the regrind circuit together with the middling froths of the Minerals Separation machines.

The tailing from the second K&K rougher and the tailing from the second Minerals Separation rougher are waste products.

Aerofloat No. 15 is fed to all grinding units by mechanical feeders; cyanimid and soda ash are mixed dry and fed by a reconditioned Merrill zinc feeder to the ore at the gyratory crusher.

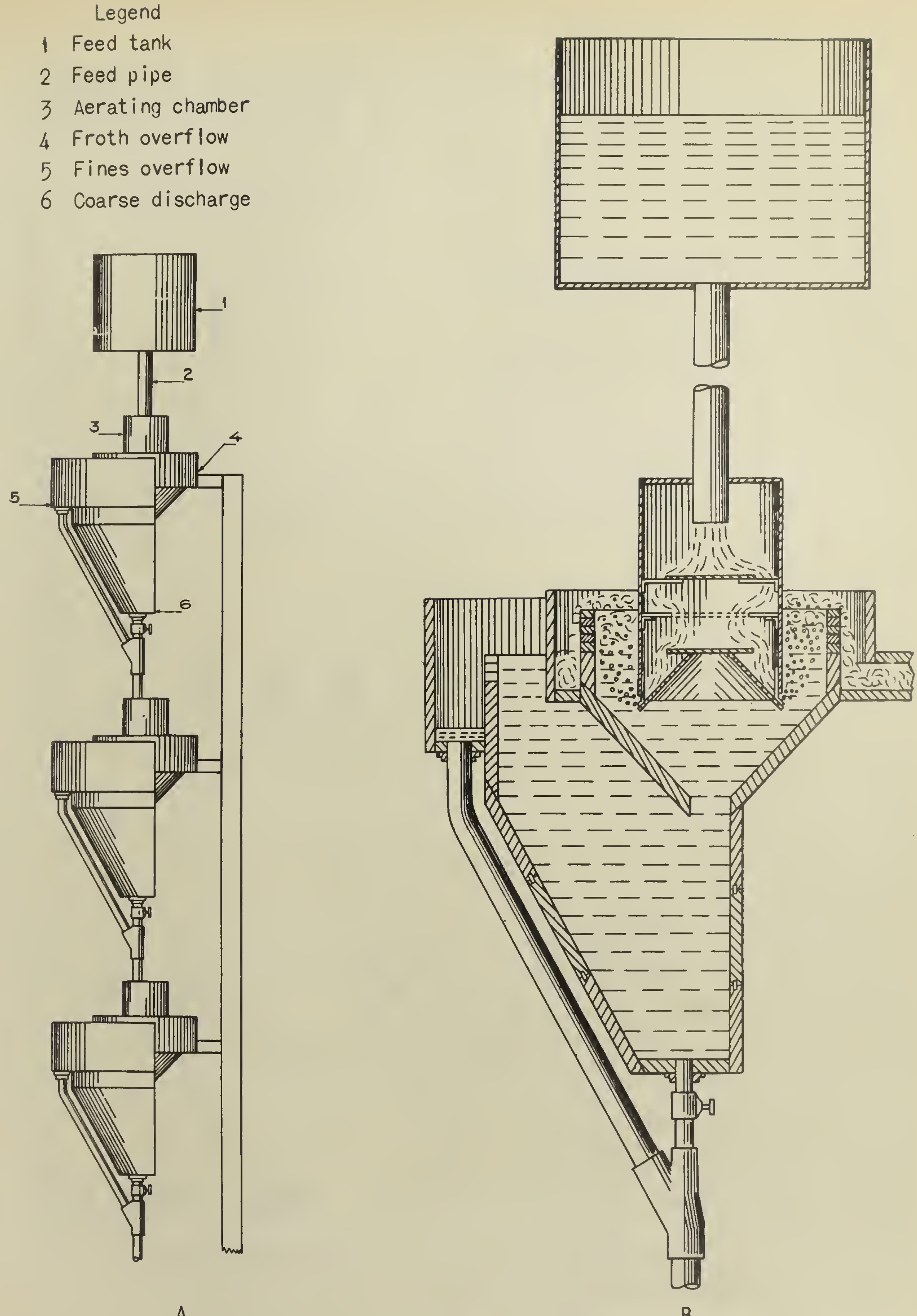
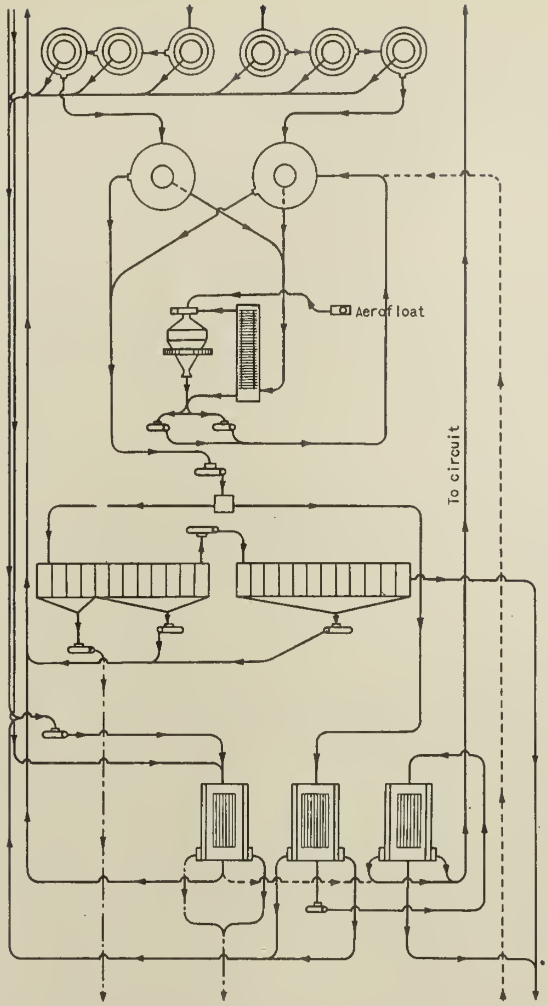


Figure 4.- A, Three-cell Premier flotation machine; B, details of Premier flotation cell



Legend

Fine grinding and Premier flotation cells

Six Premier flotation cells

Two cone classifiers

Mechanical reagent feeder

Hardinge ball mill, 6-foot by 16-inch,
27 r.p.m., 50-hp. motor, Tex-rope drive

Two Wilfley sand pumps, 2-inch, 10-hp. motors

Dorr duplex model C classifier

Flotation M.S. and K & K machines

Wilfley sand pump, 4-inch, 20-hp. motor

Surge box, 4- by 4-foot

Wilfley sand pump, 4-inch, 20-hp. motor

Two M.S. sub A flotation machines, 12-cell,
24-inch, 75-hp. motor per machine

Two Wilfley sand pumps, 2-inch, 5-hp. motor

Wilfley sand pump, 2-inch, 5-hp. motor

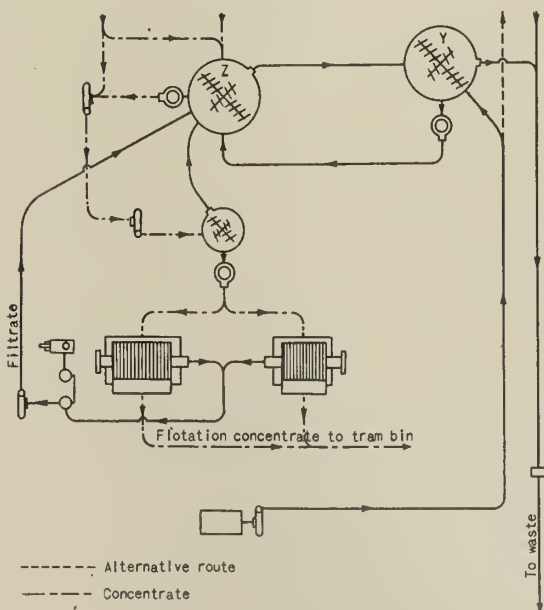
Wilfley sand pump, 2-inch, 10-hp. motor

K & K cleaner flotation cell, 12-foot,
double-spitz

Two K & K rougher flotation machines,
12-foot double-spitz

Wilfley sand pump, 2-inch, 10-hp. motor

Figure 5.- Flow sheet of flotation.



Legend

Thickening and filtering concentrate

(Y) Dorr thickener, 30-foot

Dorrco pump, No. 4

(Z) Dorr thickener, 30-foot

Wilfley sand pump, 3-inch,
25-hp. motor

Dorrco pump, No. 4

Wilfley sand pump, 2-inch
15-hp. motor

Dorr thickener, 18-foot, equipped
with 40-foot mechanism, 5-hp. motor

Dorrco pump, No. 4

Oliver filter, diameter 5 feet 4 inches,
face 4 feet

Oliver filter, diameter 5 feet 4 inches,
face 6 feet

Vacuum pumps, moisture trap

Dewatering pump

Galigher automatic tailing sampler

Sump

Krogh sump pump, 3-inch, 15-hp. motor

Figure 6.- Flow sheet of dewatering.

Table 7.- Data for primary and regrinding mills

	Primary mills		Regrinding mills		
	Marcy	Hardinge	Denver Colorado mill	Allis-Chalmers ball granulator	Hardinge
Size: diameter and length..............feet....	6 by 4 1/2	8 by 3	4 by 8	5 by 8	6 by 1.33
Speed.............................. r.p.m. ...	25	23	31	28	27
New feed per 24 hours.................. tons ...	175	290	96	127	89
Total feed per 24 hours.............. do. ...	350	510	233	414	208
Type of circuit	Closed	Closed	Closed	Closed	Closed
Installed horsepower..........................	90	150	60	75	75
Method of driving..............................	Clutch	Tex-rope	Belt	Flexible coupling	Tex-rope
Solids in mill pulps.................per cent...	79.3	76.8	76.0	75.5	76.5
Solids in classifier overflows........ do. ...	54.2	62.4	22.8	26.1	62.8
Type of feeder.................................	Scoop	Comb. drum	Scoop	Scoop	Scoop
Ball load.......................... pounds ...	7,000	33,000	11,000	17,500	10,000
Liner material.................................	Mn steel	Mn steel	Cast-iron	Cast-iron	Cast-iron
Liner consumption.................pounds per ton	0.40	0.38	0.14	0.08	0.04
Grate material.................................	C.r. steel	....	....	....	....
Grate consumptions................pounds per ton	0.07	....	....	....	....
Diameter of balls...................... inches ...	5	5	2 1/2	2 1/2	1 3/4
Ball material..................................	Forged steel	Forged steel	Cast-iron.	Cast-iron	Forged steel
Ball consumption..................pounds per ton	1.94	2.39	1.34	1.33	0.61
Type classifier................................	Dorr duplex	Dorr duplex	Premier cone	Premier cone	Premier cone and Dorr duplex
Size of classifier.....................feet.....	4.5 by 14.7	6 by 20	4 1/2	4 1/2	8 and 4 1/2 by 14.7
Speed of rakes per minute.............strokes...	27	27	....	....	18

Table 8.- Screen analyses of products of primary grinding circuits, 1931

Mesh	No. 64 1/2 Marcy						8-foot by 36-inch Hardinge					
	Ball-mill discharge		Classifier				Ball-mill discharge		Classifier			
			Return		Overflow				Return		Overflow	
	Per cent	Cum.	Per cent	Cum.	Per cent	Cum.	Per cent	Cum.	Per cent	Cum.	Per cent	Cum.
+20...	26.7	26.7	33.4	33.4	2.5	2.5	26.3	26.3	28.5	28.5	1.3	1.3
+50...	30.6	57.3	39.4	72.8	20.4	22.9	31.6	57.9	41.1	69.6	17.1	18.4
+80...	10.3	67.6	11.4	84.2	16.3	39.2	10.5	68.4	12.6	82.2	16.4	34.8
+100..	5.4	73.0	3.9	88.1	8.7	47.9	5.3	72.7	4.4	86.6	8.6	43.4
+150..	4.4	77.4	2.4	90.5	7.0	54.9	4.0	77.7	2.7	89.3	7.5	50.9
+200..	4.5	81.9	2.1	92.6	8.7	63.6	4.8	82.5	2.2	91.5	9.7	60.6
-200..	18.1	100.0	7.4	100.0	36.4	100.0	17.5	100.0	8.5	100.0	39.4	100.0

Table 9.- Screen analyses of regrinding circuit products, 1931

Screen size, mesh	Marcy-mill section, 4 1/2-foot cone classifier				4- by 8-foot regrind-mill discharge		Hardinge-mill section, 4 1/2-foot cone classifier				5- by 8-foot regrind-mill discharge	
	Discharge		Overflow				Discharge		Overflow			
	Per cent	Cum.	Per cent	Cum.	Per cent	Cum.	Per cent	Cum.	Per cent	Cum.	Per cent	Cum.
+50...	39.9	39.9	1.9	1.9	7.8	7.8	37.5	37.5	0.3	0.3	27.4	27.4
+80...	27.7	67.6	6.7	8.6	12.9	20.7	25.3	62.8	1.7	2.0	23.0	50.4
+100..	14.1	81.7	9.8	18.4	11.5	32.2	13.0	75.8	7.3	9.3	11.1	61.5
+150..	8.0	89.7	11.2	29.6	15.2	47.4	8.9	84.7	8.0	17.3	7.1	68.6
+200..	5.1	94.8	11.7	31.3	16.0	63.4	6.7	91.4	8.1	25.4	5.5	74.1
-200..	5.2	100.0	58.7	100.0	36.6	100.0	8.6	100.0	74.6	100.0	25.9	100.0

Screen size, mesh	8-foot cone classifier handling primary tailing				Dorr classifier in circuit with 6-foot Hardinge mill				6-foot Hardinge regrind-mill discharge	
	Discharge		Overflow		Return		Overflow			
	Per cent	Cum.	Per cent	Cum.	Per cent	Cum.	Per cent	Cum.	Per cent	Cum.
+50...	13.5	13.5	0.3	0.3	13.1	13.1	0.9	0.9	6.0	6.0
+80...	23.7	37.2	2.2	2.5	38.1	51.2	12.1	13.0	10.1	16.1
+100..	19.7	56.9	7.7	10.2	23.5	74.7	19.9	32.9	10.8	26.9
+150..	15.6	72.5	9.8	20.0	12.4	87.1	21.5	54.4	11.7	38.6
+200..	13.1	85.6	12.1	32.1	6.2	93.3	13.6	68.0	12.5	51.1
-200..	14.4	100.0	67.9	100.0	6.7	100.0	32.0	100.0	48.9	100.0

Table 10.- Consumption and cost of balls used in grinding circuits, years 1927-1931, inclusive

Year	Ore milled, dry tons	Total steel balls used, pounds	Average unit cost of balls per 100 pounds	Total cost of balls	Balls consumed per ton of ore, pounds	Cost of balls per ton of ore treated
1927	147,536	595,635	$3.76	$23,061.47	4.03	$0.156
1928	162,112	720,625	3.76	27,085.94	4.44	.167
1929	165,143	668,540	3.47	23.215.62	4.05	.141
1930	151,936	668,750	3.99	26,664.27	4.40	.175
1931	169,760	707,844	3.29	23,271.83	4.17	.137

Table 11.- Consumption and cost of ball mill liners, years 1927-1931, inclusive

Year	Ore milled, dry tons	Total cast-iron and steel liners used, pounds	Average unit cost of liners per 100 pounds	Total cost of liners	Pounds per ton milled	Liner cost per ton of ore milled
1927	147,536	94,240	$13.30	$12,531.74	0.639	$0.085
1928	162,112	104,760	12.19	12,767.93	.646	.079
1929	165,143	93,289	11.02	10,278.12	.565	.062
1930	151,936	112,745	10.71	12,072.56	.742	.079
1931	169,760	89,286	10.32	9,218.54	.526	.054

The flow sheet as described employs a stage grinding-flotation-classification system, which gives a differential grinding effect whereby the heavier sulphide particles are ground much finer than the gangue material. The development of this flow sheet resulted from data obtained from mill operations and laboratory tests. It was found that 50 to 60 percent of the mineral values could be recovered when the ore was ground 50 percent minus 200 mesh, and that the balance could be removed as a middling product to be returned to the circuit for further classifying, grinding, and flotation. When the present method was adopted, the size of grinding was decreased from 81 percent minus 200 mesh to 70 percent minus 200 mesh without affecting either recoveries or grade of concentrate.

A 3-cell Premier flotation machine is shown in figure 4A, and the details of one cell are shown in figure 4B. This machine is of the cascade type and was developed for use between the primary grinding and regrinding circuits and in this position removes a concentrate as soon as grinding has reached the degree where flotation is possible. In addition to the advantage of permitting coarser grinding, the Premier flotation machine removes the valuable minerals from the circuit as soon as possible, eliminates the peaks in the feed to the mechanical flotation machines, and produces a coarse concentrate, which in turn improves thickening and filtering operations. Previous to the installation of those cells, when flotation feed was ground 81 percent minus 200 mesh, the thickening and filtering capacity was taxed to the limit in handling the finely ground concentrate. After the introduction of the stage grinding-flotation-classification system, the percent of minus 200-mesh material in the concentrate decreased from 88 to 60 percent and the capacity of the filters increased from 525 to 998 pounds per square foot per 24 hours.

Typical analyses of flotation products and typical recoveries of gold and silver are given in table 12; typical screen-assay analyses of final concentrate and tailing are given in table 13.

Table 12.- Typical analyses of flotation products and recoveries for August, 1932

	Weights			Analyses						Recoveries	
	Percent of total	Moisture, percent	Dry tons	Ounces per ton: Gold	Ounces per ton: Silver	Percent: Lead	Percent: Zinc	Percent: Iron	Percent: Insol.	Percent gold	Percent silver
Heads	100.0	2.0	12,782	0.35	6.55	0.7	1.7	6.0	74.2	.....	.....
Primary concentrate from Premier cells	.....	.....	......	2.48	34.00	2.4	7.7	23.5	10.8	.....	.....
K&K cleaned concentrate.	.....	.....	......	2.92	51.12	2.7	8.8	27.3	13.0	.....	.....
Feed to secondary flotation machines.....	.....	.....	......	.17	4.52	.2	1.9	5.1	79.5	.....	.....
M.S. Concentrate	.....	.....	......	2.80	60.50	3.1	10.5	29.8	7.6	.....	.....
Middlings ...	.....	.....	......	.62	18.67	1.2	9.1	18.3	39.0	.....	.....
Final concentrate	11.58	7.38	1,479	2.82	53.73	4.3	10.3	31.7	9.6	96.67	87.71
Final mill tailing	88.42	76.4	11,303	.013	0.91	.2	.6	2.6	82.6	3.33	12.29

Table 13.- Screen-assay analyses of final concentrate and tailing, August, 1932

	Plus 150-mesh			Plus 200-mesh			Minus 200-mesh		
	Weight, per-cent	Assay, ounces per ton		Weight, per-cent	Assay, ounces per ton		Weight, per-cent	Assay, ounces per ton	
		Gold	Silver		Gold	Silver		Gold	Silver
Final concentrate.	29.6	4.47	39.26	11.4	2.96	33.92	59.0	1.92	64.80
Tailing...	12.9	.021	.98	12.6	.022	.90	74.5	.013	.81

DEWATERING OF CONCENTRATE

Referring to the flow sheet of dewatering of concentrate given in figure 6, all concentrates are collected in one 30-foot Dorr thickener. The thickened pulp is delivered by a No. 4 Dorrco simplex pump to a pump feed box and from there it is elevated in two stages by 3-inch and 2-inch Wilfley pumps to an 18-foot thickener located above the tram terminal bins. Lime is used to aid settling and is added to the pulp at the rate of 0.3 pound per ton of concentrate.

The thickened pulp from the 18-foot thickener is delivered to either of two Oliver filters, 5 feet 4 inches by 6 feet or 5 feet 4 inches by 4 feet in size. The filters are so situated that the cakes, when discharging from the drums, drop into the concentrate bin and from there are drawn by gravity into buckets and conveyed by aerial tramway to the bunkers at the dock.

A flapping mechanism, shown in figure 7, is used to aid filtering. Before this mechanism was installed on the filters it was noted that a film of extremely fine material formed on the surface of the cake when the drum rotated through the pulp in the hopper, thus restricting the passage of air through the cake and retarding the removal of moisture. The use of the flapping mechanism broke this film and reduced moisture content from 11.3 to 10.0 percent. The production of coarse cleaned primary concentrate further reduced the moisture content from 10.0 to 8.3 percent. Filtering data are presented in table 14.

Table 14.- Filtering data

	Filter 1	Filter 2
Size, feet	5.33 by 4	5.33 by 6
Vacuum, inches of mercury at 1,400 feet elevation	21	21
Kind of cloth	Palma twill	Palma twill
Average life of cloth .... days......	113	108
Average duty per 24 hours .... tons......	31	45
Solids in feed pulp.................. percent...	81.3	81.3
Moisture in cake..................... do.	8.30	8.25

METALLURGICAL DATA

Metallurgical data for the year 1931 are given in table 15.

Table 15.- Metallurgical data, 1931

Assays of heads:		
Gold	ounces	0.305
Silver	do	5.93
Ore treated	tons	169,760.52
Days operated:		
Marcy unit		345.80
Hardinge unit		346.76
Operating time:		
Marcy unit	percent	95.00
Hardinge unit	do	95.26
Average ore milled per 24 hours:		
Marcy unit	tons	176.35
Hardinge unit	do	290.02
Total	do	466.37
Recovery of gold	percent	93.35
Recovery of silver	do	86.29
Concentration ratio, tons into 1		8.44
Moisture in concentrate	percent	8.28
Net water consumption per ton of ore milled	imperial gallons	700.
Ball consumption per ton of ore milled	pounds	4.17
Liner consumption per ton of ore milled	do	.526
Consumption of reagents per ton of ore milled:		
Aerofloat	pounds	.280
Sodium carbonate	do	.030
Cyanide (cyanamid)	do	.012

SAMPLING

The sample of mill heads is taken by Galigher automatic samplers which cut the overflow pulps of the primary classifiers at 7-minute intervals. An accurate sample of the ore actually milled is obtained by this method, as the grinding has been sufficiently fine to permit of accurate sampling.

The main tailing launder is equipped with a similar automatic sampler which takes the tailing sample, cutting the pulp stream at 7-minute intervals.

The grab samples of shipping concentrate are taken during the loading of each tramway bucket. Each bucket carries approximately 1,050 pounds. Over a period of years this method of sampling has yielded results which have checked smelter assays remarkably closely.

All intermediate concentrator products are sampled by hand.

All samples are for 24-hour periods and are collected at 10 p.m. Assay results from these samples are available the following day at 1:30 p.m.

EXPERIMENTAL LABORATORY

An experimental laboratory is maintained in connection with the milling operations.

WATER SUPPLY

The fresh-water supply is obtained entirely from the mine. The water flows from the upper mine workings to the mill supply tank. The mine water contains no substance injurious to flotation, and has a pH value of 7.2. The approximate yearly average temperature of this water is 36°.

LABOR

Few changes occur in the personnel of the operating crew, with the result that the men have become very familiar by observation with the changes in the ore passing through the plant. Operators are able to make necessary changes to take care of these fluctuations, in time to avoid disruptions in the circuits. Observations from panning various mill products or the use of "pilot" tables are not dependable as guides to flotation efficiency on Premier ore.

Milling operations are in charge of the mill superintendent, who is directly responsible to the management. The assistant superintendent is in charge of day-shift operations. A shift boss has charge of operations on afternoon and night shifts.

The repair crew is in charge of a repair foreman, who with two repairmen, take care of all routine repair work. One of the repairmen, who is an electrician, looks after minor electrical repairs and motor inspection. All major repairs are turned over to the mechanical department.

There are 25 men employed in the mill. Positions and rates of pay are listed in table 16.

Table 16.- Labor data

Position	Number of men	Rate per hour
Superintendent	1	Salary
Shift bosses	3	$0.78
Jaw crusher	3	.56
Gyratory crusher	3	.48
Ball mill	3	.61
Flotation	3	.56
Filter	3	.56
Repair foreman	1	.78
Repairmen	2	.56
Laborers	1	.54
Conveyormen	1	.48
Testing department		Salary

SUPPLIES

Supplies are issued as required by the warehouse department upon requisitions signed by the mill superintendent. These requisitions, after passing through the warehouse, are checked daily by the management and returned to the mill superintendent.

On account of the distance from the source of supply for repair parts, reagents, etc., it is necessary to anticipate mill requirements from 1 to 3 months in advance.

CLIMATE

As the climate is comparatively mild, very little heating in the mill building is necessary. Owing to the occasional extremely heavy snowfall, however, very strong building construction is required.

Weather reports for the years 1925 to 1931, inclusive, are shown in table 17.

Table 17.- Weather reports, years 1925-1931

Year	Temperatures, °F.			Precipitation, inches	
	Maximum	Minimum	Mean	Snowfall	Rainfall
1925	86	-1.0	42.5	559	31.16
1926	82	1.0	41.0	434	51.63
1927	86	-6.0	37.0	555	30.00
1928	81.5	9.0	41.0	492	35.66
1929	78	0.0	41.0	345	46.00
1930	81	2.0	41.4	499	49.82
1931	85	4.0	41.0	433	48.27

POWER

All machinery is run by motors, power for which is supplied from the company's own plant. This plant is situated on Cascade Creek, half a mile below the mill. The plant consists of six Fairbanks-Morse Diesel engines with a total combined capacity of 1,680 hp.; one 3-runner, low-head Pelton wheel of 550 hp.; and two single-runner high-head Pelton wheels with a combined capacity of 550 hp.

During the summer months power is generated by the hydroelectric plant; and during the winter, owing to the lack of water, the Diesel engines are used. Power is generated and transmitted at 2,300 volts and is stepped down for some of the motors to 440, and for lighting to 110 volts.

A summary of power consumption and distribution for the year 1931 is presented in table 18.

Table 18.- Summary of power consumption for 1931 (169,760 tons of ore treated)

	Kilowatt-hours	Kilowatt-hours per ton	Percent of total power
Primary breaking	236,444	1.393	4.46
Secondary breaking	396,027	2.333	7.47
Primary grinding	1,211,681	7.138	22.84
Secondary grinding	972,008	5.726	18.32
Classification	76,562	.451	1.44
Flotation	1,038,584	6.118	19.58
Filtering	213,044	1.255	4.02
Pumps	991,980	5.843	18.70
Lighting	168,120	.990	3.17
Totals	5,304,450	31.247	100.00

COSTS

A summary of milling costs for the year 1931 is presented in table 19.

Table 19.- Summary of milling costs for 1931 (169,760 tons of ore treated; 48,389 ounces of gold and 868,371 ounces of silver produced)

	Operating labor	Power	Supplies	Repairs		Total
				Labor	Supplies	
Primary crushing	$0.028	$0.013	....	$0.024	$0.027	$0.092
Secondary crushing	.030	.019	....	.008	.013	.070
Primary grinding	......	.070	1/$0.137	.026	.104	.337
Secondary grinding	1/.035	.055	....	.010	.018	.118
Classifying	......	.004	....	.003	.003	.010
Flotation	.032	.050	....	.020	.019	.121
Reagents	......	......	.059	......	......	.059
Filtering	.033	.010	....	.004	.004	.051
Pumps, pipe lines, and launders	......	.047	....	.013	.030	.090
Oils, waste, and grease	......	......	.009	......	......	.009
Small tools	......	......	.002	......	......	.002
Motors, wiring, and lights	......	......	....	.007	.006	.013
Lighting	......	.006	.002	......	......	.008
Heating	......	......	.008	......	......	.008
Experimental	.009	......	....	......	......	.009
Building repairs	......	......	....	.003	.003	.006
Superintendence	.064	......	....	......	......	.064
Total	0.231	0.274	0.217	0.118	0.227	1.067

1/ Primary grinding, secondary grinding, and classifying.

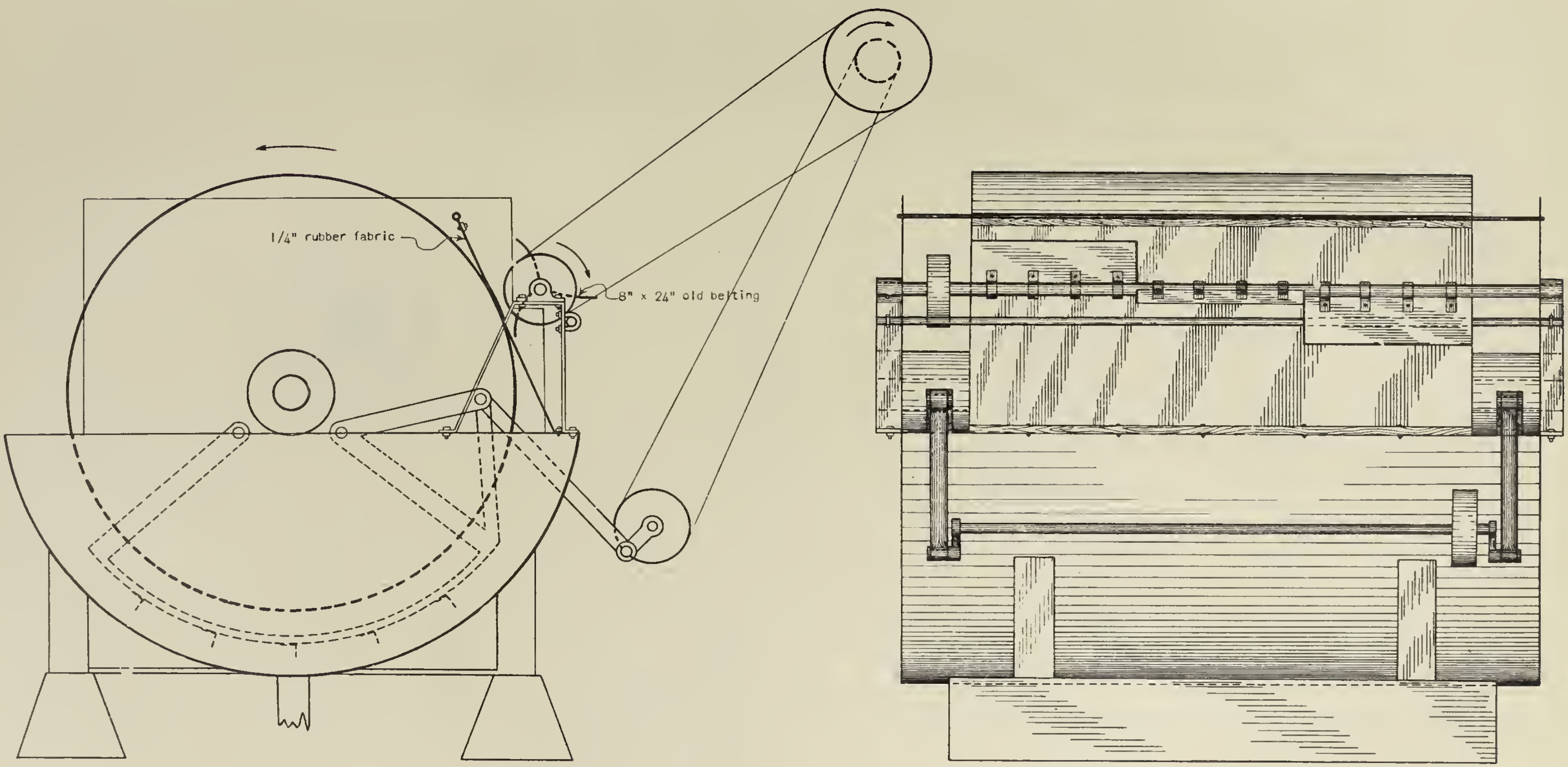


Figure 7.- General arrangement of Oliver filter, showing rubber flappers for breaking film on filter cake.

I. C. 6745

September, 1933

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

ABOUT HELIUM

BY

Andrew Stewart

INFORMATION CIRCULAR

UNITED STATES BUREAU OF MINES

ABOUT HELIUM[1]

By Andrew Stewart[2]

ACKNOWLEDGMENTS

The author desires to express appreciation to R. A. Cattell, C. W. Seibel, and H. S. Kennedy, of this Bureau, and to Dr. Richard Wiebe of the fixed nitrogen research laboratory, Department of Agriculture, for helpful criticism and comments in the preparation of this paper. More especially thanks are due to R. A. Cattell for his many valuable constructive suggestions.

INTRODUCTION

From time to time, the Bureau of Mines receives inquiries from the non-technical public on the subject of helium. People ask for information concerning its properties; they want to learn whence it is derived, how it is produced, and to what uses it may be put. In consequence, the following paper has been written to give in simple terms an account of this most remarkable of gases. It has been thought desirable for purposes of analogy to include reference to all the members of the group of elements to which helium belongs, and, incidentally, to embody in the narrative certain other information pertinent to the subject.

Beginning the Story

The story of helium is replete with interest and is unique in the annals of science. The imagination is stirred as one learns how this wonder element, which up to as recently as 15 years ago was obtainable only in very small quantities and at a fabulous cost of the order of $2,500 per cubic foot, is now being produced day by day for about 1 cent per foot and in volumes vast enough to float great airships. One is fascinated as the narrative unfolds concerning the discovery of helium, first as revealed in the gases surrounding the sun, and then as a part of the substances of the earth, and it is learned that this element is actually brought forth by some subtle wizardry of Nature, involving transformation or transmutation of one elemental substance into another.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6745."

2 Senior chemist, helium division, U. S. Bureau of Mines.

Transmutation of Elements--The Alchemists--Molecule and Atom

Transmutation of element into element was once but the chimerical dream of the alchemists, those ancient mystery workers, who, from early Egyptian times down through the gloom enveloping all things intellectual in the Middle Ages, strove in vain to seek out a certain magical agent--the "philosopher's stone," or "elixir"--with which they believed it would be possible, according to a concept of the essential unity of all matter, to resolve substance into substance, more especially, however, baser metals into gold, whereby there would be opened up a source of limitless riches. After them came the investigators of later generations; men who viewed matter as divided into minute masses which they called "molecules", a word from the Latin meaning "little masses." This was no new concept, however. The theory that matter is made up of minute particles had already been advanced by the greatest of Greek physical philosophers, Democritus, and was maintained by him and his followers in the fifth century B. C. But with the passage of time this was forgotten. Molecules represent the limit of physical division, a division not affecting the inherent nature of the substance. A further division is the atom. This represented, initially, the ultimate limit of chemical, or fundamental, mutation, the word "atom" being derived from the Greek and meaning "indivisible."

Up to a comparatively recent day, the succession of investigators following the alchemists laughed to scorn the possibility of transmutation of elements. It was believed that the atom, the soul of the element, was impervious to change. But with the passage of time, bringing enlightenment through research, it has become known that atoms are by no means the limit of material division. Far from it. It has been conclusively shown that atoms are more or less intricate complexes, or systems, of electrical particles (1),[1] and, working on this basis, scientists have been able to cause them to undergo change in structure, giving rise to resolution of one elemental substance into another. Moreover, it has been found that such change occurs even spontaneously in so-called "radioactive" mineral substances, to which reference will be made later.

Thus, strangely, the dream of the ancient alchemist, once considered fantastic, has at last come true. However, the artificial transmutation of element into element has as yet been effected on a scale so small as to be only of scientific interest. It remains for the future to disclose any practical benefit to mankind that may develop from further researches in this field.

The Sun, Birthplace of Helium--Telescope and Spectroscope

From time immemorial the sun, in its majesty, has been an object of man's adoration and wonder. Gradually, however, through the ages, an urge of practical curiosity concerning it began to materialize. The desire to know something of the true nature of the sun took hold on men's minds. But this did not begin to bear real fruit, in the light of scientific investigations, until the illustrious Italian savant, Galileo, in 1609, made his improvement of the

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1 Numbers in parentheses, unless otherwise obviously significant, allude to the list of references at the end of this paper.

telescope, which had been invented in Holland in 1608 (2c). The advent of the telescope, so named by Galileo, was one of the most momentous and interesting events in the scientific development of the world. It furnished the key to unlock the mystery of the heavens and was eagerly grasped and employed by such men as Galileo, Kepler, Harriot, and the distinguished astronomers who have followed after them (3, pp. 6-8).

Among the many subjects for investigation concerning the sun that agitated scientific thought after the advent of the telescope, there were two upon which attention became specially centered, and which, incidentally, led to the discovery of helium. Observed in the telescope were two remarkable solar phenomena: The so-called "sun spots", or dark-appearing areas on the sun's surface, and the fiery prominences or protuberances appearing around the periphery of the sun in the course of an eclipse.

Lockyer--Pioneer in Spectroscopic Solar Observations

A pioneer in interpreting these phenomena was J. Norman Lockyer (later Sir J. Norman Lockyer), the brilliant English astrophysicist. It was he who was the first to employ an instrument, known as the spectroscope, for investigating the physical constitution of the sun, his investigations beginning March 4, 1866 (3, pp. 117, 210 and 436; and 9). Owing primarily to his development of the spectroscope and the art of spectroscopic investigation which he devised and applied to astronomy, the sun spots are now recognized as vast vortices having huge magnetic centers extending over areas of as much as millions of square miles (4, p. 254), while the prominences or protuberances have been revealed as spectacular masses of incandescent gas, mainly hydrogen, forming a part of the gaseous envelope about the sun (3, pp. 125-126 and 390-404). They have been observed to rise from the sun's periphery to a distance of over 500,000 miles (2 d).

The Solar Eclipse of 1868

In 1868 there occurred a total eclipse of the sun. Its shadow included India in its path, and, as it was thought that the eclipse could be seen under particularly advantageous conditions in India, eminent scientists from various parts of Europe made a pilgrimage to that region to have this phenomenon under special observation (3, pp. 119 and 213). As a feature of this activity, the spectroscope was used for the first time to study, under eclipse conditions, the sun's atmosphere, which Lockyer (as suggested by his friend Dr. Sharpey, Secretary of the Royal Society) later called the "chromosphere" (3, pp. 129, 130, and 453). The existence of a continuous stratum or envelope of atmosphere surrounding the sun had been previously suspected by others, but Lockyer was not aware of this. He definitely established it spectroscopically on November 5, 1868 (5). He estimated its average thickness at about 5,000 miles. By the sun's eclipse, when its direct and blinding rays were obscured, this atmosphere was thrown into relief, presenting an ideal, if fleeting, condition for its inspection, together with the magnificiently red- and violet-colored prominences which were observed in all their fiery glory extending far into space. But, as will be

referred to later, Lockyer, following a method of his own, made his spectroscopic observations in England, in the absence of an eclipse.

The Composition of Light

To digress a moment, daylight, or so-called "white light", is really a composition of various colored lights emitted by the incandescent matter of the sun. This was discovered by Sir Isaac Newton 260 years ago. The sun's substance is now considered to be gaseous, existing, however, under incredibly high conditions of temperature and pressure. Physicists estimate that at its center the sun has a temperature of the order of 29,000,000° C., a pressure of 36,000,000,000 atmospheres, and a density approximately 28 times that of water (4, p. 259).

The Spectrum

When a beam of sunlight is passed through a glass prism, the beam is bent, or refracted, from its straight path of travel. But this is by no means all that happens. Since each of the colors composing the light is refracted to a different degree, the prism separates, or disperses, the beam into its components and their colors then become apparent. Therefore, when the refracted beam is caught on a screen it appears as a luminous, colored ribbon or rainbow of light varying in sequence from red, through orange, yellow, green, and blue to violet. This ribbon or band of light is called a "spectrum." This is a Latin word meaning "an appearance", "an image."

The Spectroscope--Fraunhofer Lines

A spectroscope is an instrument, appropriately mounted, having as its essentials a prism, or a number of prisms in series; also, lenses fixed in tubes adjusted respectively for reception and transmission of light from its source to the prism, or prisms in train, and thence, after refraction and dispersion, to the eye, in place of a screen. The light coming to the spectroscope is admitted through a narrow slit-opening in the end of the receiving tube (3, pp. 154-168; 2 b).

Observation of the spectrum of sunlight, the solar spectrum as it is called, through the spectroscope, will show that its bright ribbon of colors is not continuous. It is crossed by innumerable fine dark lines, as first observed by Dr. Wollaston, in 1802. The lines were made visible by his introduction of the narrow slit in the spectroscope to admit the light to the prism, thus establishing the basis of all the modern work which has been done with that instrument. The dark lines were mapped to the number of 576 by Joseph von Fraunhofer (3, pp. 148-152), a celebrated German optician and physicist, in 1814, since when they have been known as "Fraunhofer lines." Many thousand such lines have now been distinguished. This shows that some of the sun's rays have been obstructed on their way to the earth, as will be referred to later.

Spectra of Solids, Liquids, Gases

Now, when the light from a solid or liquid body, made sufficiently incandescent below its vaporization point (as a piece of white-hot platinum or lime), is passed through a spectroscope, the spectrum formed is a continuous ribbon of succeeding colors of light. It contains no dark lines. But, when a gas or a vapor of a substance under pressure not greatly above atmospheric, or not greatly increased in density, is made incandescent and the light emitted from it is passed through the spectroscope, the resulting refracted light does not give the continuous ribbon; instead, there are visible only some transverse lines of various colors of light appearing at different places along the path of what, in the case of the spectrum of a solid or a liquid, would be the ribbon containing the gradations of the colors of the full spectrum. The spaces intervening between the lines appear black. This is called a bright line spectrum, or, when the lines lie closely crowded together so as to form one or more bands, a band spectrum. A succession of bands with well-marked heads (or specially crowded line aggregates) is called a fluted spectrum.

Spectrum Analysis

In the more complete spectroscopes there is provided either a special adjustment of the observation tube, or a permanently fixed, specially illumined scale, for establishing the exact location of the lines of light. It has been found that each gaseous substance has its different, distinctive lines, which appear at invariably the same respective places in the spectrum. A method for analyzing substances is thereby provided. It is known as spectrum analysis. The science of spectrum analysis was founded by Kirchhoff, in collaboration with Bunsen, in 1859 (2b). By means of it elementary substances can be detected by the spectroscope in amazingly minute traces. For instance (6) a salt of sodium has been recognized by the spectroscope in the almost unbelievably small amount of less than 0.0000003 milligram ($\frac{1}{216,000,000}$ grain).

A spectroscope with attachment for accommodating photographic plates with which to photograph the spectra, thus making permanent records, is called a spectrograph (7).

The Gas Spectrum

In order to examine a gaseous substance with the spectroscope, so-called "vacuum tubes" made of glass are used. Through each end of these, electrical terminals of platinum and aluminum are sealed. Such tubes are called Geissler, or Plücker, tubes, named for their respective inventors. The vacuum tube is filled with the gas to be analyzed; the gas is then exhausted from the tube, with a pump, until there remains only a very small amount under proportionately reduced pressure. The tube is then sealed by fusing the glass with a blowpipe at the opening where the gas was admitted, and the electrical terminals are connected to a high-tension current, whereupon the rarified gas in the tube becomes highly luminous. Helium gives a brilliant glow ranging from green to canary yellow, with increasing pressure in the tube. The well-known

brilliant signs for advertising purposes now seen on every street consist of such vacuum tubes, on a more or less pretentious scale, containing various gases. The signs showing the gorgeous crimson light contain neon. Almost any color and tone can be produced by employing certain gases and combinations of these in plain or colored glass tubes (8).

Nature of Fraunhofer Lines

The light from the body of the sun would be seen in the spectroscope without the dark Fraunhofer lines if it flowed to the earth without hindrance, since incandescent gaseous bodies when under great pressure or in very thick volumes, as found in the stars, give continuous spectra just as luminous solids and liquids (2d). However, this light must pass through the atmosphere surrounding the sun and our own atmosphere. The gases of the sun's atmosphere give, of course, bright-line spectra when observed alone, but surrounding the sun they form an impediment to those very rays of light emitted from the main solar body which they themselves give out when in a state of incandescence. The positions in the spectrum which their rays occupy, therefore, appear comparatively dark. Moreover, some gases and vapors in the earth's atmosphere impede certain portions of the sun's light. The theory is that the radiating energy of the rays from the incandescent matter of the sun is absorbed, or used up, in setting in vibration the atoms of the comparatively cooler gases in the solar atmosphere and the gases of our own air through which the rays are projected; therefore, these rays do not get through to the spectroscope, or do so in a greatly weakened state. In consequence, the solar spectrum is called an absorption spectrum.

An analogy is seen in the realm of sound. For instance, if a musical note is struck in the presence of a stretched piano wire, which when set in vibration itself would give forth that note, the struck note will set the wire in vibration. In the same way, when a light encounters a gas which when luminous would give off the same vibration as the light, then the light sets the atoms of the gas in vibration, but its force is thereby weakened or quenched; hence dark or comparatively less luminous lines attributable to it appear in the spectrum instead of light lines. This can be demonstrated by interposing sodium vapors made incandescent in an alcohol flame, between a lime light and the spectroscope. A part of the bright yellow rays given off by the intensely heated lime, corresponding to the vibrations of the sodium spectrum, will be absorbed and appear black, instead of being made more brilliantly luminous by the yellow light from the comparatively cooler and therefore less brilliant sodium vapors in the flame.

There are also many substances, for instance, oxygen, water and iodine vapors, nitrogen peroxide, and solutions of blood and of aniline dyes, which at ordinary temperatures show characteristic selective absorption on passage of light from a white-hot solid through them, and this circumstance provides a further means for spectrum analysis--detection of substances through lines of absorption.

Stokes, Ångström, Stewart, and Kirchhoff

Light-absorption phenomena were first explained by Prof. (subsequently Sir) George G. Stokes, of Cambridge University, about 1852. This was followed by a further explanation by Ångström. The law which connects radiation and absorption of heat rays was discovered and experimentally proved by Balfour Stewart in 1859. Stewart generalized his conclusions for all rays. In the same year the great German physicist, Kirchhoff, established experimentally the same law for light rays (3, pp. 185 to 195). From what has been said, it will be realized that, to use Lockyer's expression, "the light proceeding from any substance contains, as it were, an autobiography of that substance in a strange language, certainly, but one capable of being translated into the vulgar tongue by passing a light through a prism."

DISCOVERY OF HELIUM

Lockyer's Discovery of Nature of Chromosphere, Sun Prominences.

As has been said, J. Norman Lockyer, back in 1866, conceived the idea of examining the light of the sun spots and of the solar prominences with the spectroscope (3, pp. 117, 210, and 436; and 9). He conceived that if bright lines were revealed by the instrument in the spectrum of the prominences, not only would their gaseous nature be shown, but it would probably be possible to determine of what gases they were composed. However, he found that the spectroscope then available to him did not possess sufficient dispersive power to overcome the exceeding brightness of the body of the sun, which drowned out all subordinate lights.

It should be stated that Lockyer, in pursuing his astronomical work, encountered many serious obstacles, prominent among them being lack of funds, and it was not until October 1868 that this great astrophysicist was in possession of a specially constructed instrument suitable for his purpose. With it attached to his telescope, using this method for solar investigation which he had originated in making his observations in 1866 (3, pp. 210 and 436; 4, p. 233), he conclusively showed on October 20, 1868 (10 and 5), that the prominences did indeed give a bright-line spectrum, and were, therefore, immense volumes of gas, shooting out from a solar envelope of such gas; furthermore, they were composed in part, at least, of hydrogen (3, pp. 125-126; 5).

Lockyer Observes and Fixes Position of Yellow Line

Lockyer also noted a line in the yellow part of the spectrum more refrangible than the two already known D lines referable to sodium, and this line he fixed on November 15, 1868, at a point called D_3 (5). For some time Lockyer and Prof. (later Sir) Edward Frankland (11; and 4, pp. 41 and 238), the distinguished English chemist, who was interested in investigating hydrogen and during a certain period cooperated with Lockyer, putting at his disposition the facilities of his laboratory, thought this line might be due to hydrogen, visible under special solar conditions of temperature and pressure.

Lockyer Ascribes Yellow Line to Unknown Substance Which He Calls "Helium"

After exhaustive experimentation in an endeavor to produce the line with hydrogen, Lockyer became convinced that it was due to some substance in the sun then unknown on earth. He coined the name "helium" for this substance, in the first instance for laboratory use, deriving the word from "helios", the Greek name for the sun (4, pp. 42 and 274). He was not among those who journeyed to India to observe the eclipse. His spectroscopic observations of the sun were made in England in full daylight. The method he devised made it unnecessary to wait for an eclipse in order to study the prominences.

In his presidential address to the Vesey Club, November 12, 1895, Lockyer gave an account of the discovery of helium, which was printed in the English scientific journal, Nature, of February 6 and 13, 1896. In this address many of the historic facts given herein find mention. Lockyer founded Nature in 1869, and was its editor for 50 years, until 1919, the year before he died.

Observers of the Eclipse in India

Several of the observers of the eclipse in India in 1868, among them the distinguished French astonomer, Janssen, noted a number of bright lines in the spectrum of the solar chromosphere and thus recognized the gaseous nature of the prominences, but they were uncertain as to the exact position of the lines they saw in the spectrum. Captain Herschel, Major Tennant, and M. Rayet each saw varying numbers of lines, one reported as corresponding to D (3, pp. 119-125; 5; and 12).

Janssen's Observations

Janssen made his observations at first during the fleeting moments of the eclipse on August 18, 1868, and then, beginning the next day, for 17 days thereafter he observed the unobscured sun, following a method similar to that which had been proposed by Lockyer in 1866.

Reports of Lockyer and Janssen to French Academy

Janssen reported his findings by telegram to the French Academy of Sciences (12). But he did not observe the D_3 line at that time (3, p. 128), or if he did, it failed to make any special impression upon him. He did not mention it in his telegram, nor in the report that followed it. Lockyer, working in England, on October 20, and observing the same phenomenon concerning the prominences, reported his findings that day to the Royal Society (10). Announcement of his discovery was also made without delay to the French Academy by Warren De la Rue, introducing two letters, one from Balfour Stewart, dated October 21, and one from Lockyer, dated October 23, both addressed to himself. Strangely enough, owing to the distance between India and France and the comparative slowness of travel in those days, the announcement by De la Rue of Lockyer's discovery, and Janssen's report by letter to the Academy (13), were received on the same day only a few minutes apart (14; and 3, pp. 440 and 442).

Credit for Discovery of Bright Lines in Protuberances

Here was presented a question of proper apportionment of credit for the discovery of the gaseous nature of the prominences. Janssen had actually observed the bright lines in the spectrum of the chromosphere two months prior to Lockyer, but it was Lockyer who, over $2\frac{1}{2}$ years before that, had conceived the idea of investigating the protuberances with the spectroscope and who had originated the method for doing it. He was prevented from making the actual discovery many months earlier than Janssen solely by difficulties beyond his control which kept him from securing a spectroscope of sufficient power for the purpose. However, this excited no feeling of rivalry or jealousy between the two men. On the other hand, it served to cement a sincere and close friendship between them which lasted during the remainder of their lives. The French Academy had a medallion struck in 1868 (4, pp. 40, 41, and 205), commemorating the great discovery. It bears the relief profiles and names of both Janssen and Lockyer, and the inscription: "Analyse des protuberances solaires. 18 aout 1868." (Analysis of the sun's protuberances. August 18, 1868). Janssen died in December 1907. Lockyer died, in his 85th year, on August 16, 1920.

Father Secchi's Observations

The Jesuit priest, Father Angelo Secchi, director of the observatory of the College of Rome, and recognized as one of the world's foremost astronomers, had made spectroscopic observations of the sun at Rome. He verified Lockyer's observation of the new yellow line and was aware that it did not exactly coincide with the double sodium line, being more refrangible and hence nearer the green area of the spectrum (3, p. 461; 5; 15). Father Secchi was born in 1818 and died in 1878. It is interesting to learn that he was not a stranger to the United States, since at one time he was on the faculty of Georgetown University, at Washington, D. C. Lockyer also was personally known here. He was a visitor to the United States in 1878, as an observer of the total eclipse of the sun of that year.

Dr. Hillebrand Almost Discovered Helium in a Mineral

In 1891, Dr. W. F. Hillebrand, then of the U. S. Geological Survey, reported that he had obtained an inert gas by heating uraninite (a mineral containing uranium and thorium) with weak sulphuric acid, in connection with a series of analyses of varieties of this mineral (16). He took this gas to be nitrogen, since it gave chemical reactions attributable to that element, such as evolving nitrous fumes when it was mixed with oxygen and subjected to the action of an electric current discharge, or "sparking," as it is called, and forming ammonium chloride on sparking in the presence of hydrogen and hydrochloric acid. Furthermore, this gas gave the brilliant, fluted nitrogen spectrum. Of course, these reactions proved that nitrogen was present. This won renown for Hillebrand as the first to discover free nitrogen gas in a mineral; or rather, as Hillebrand said, "in the primitive crust of the earth."

However, Hillebrand found the spectrum of this gas to be somewhat unusual for nitrogen; it contained some strange lines not identifiable with any of the known, mapped ones. Unfortunately, he carried the investigation no further. Failing to do so cost him the distinction of discovering helium on earth. He, himself, said in a letter to Professor Ramsay (17, pp. 56-57; 22), who subsequently made the discovery:

> The circumstances and conditions under which my work was done were unfavorable; the chemical investigation had consumed a vast amount of time, and I felt strong scruples about taking more from regular routine work. I was a novice at spectroscopic work of this kind.... It doubtless has appeared incomprehensible to you, in view of the bright argon and other lines noticed by you in the gas from cleveite, that they should have escaped my observation. They did not. Both Dr. Hallock and I observed numerous bright lines on one or two occasions, some of which could be accounted for by known elements--as mercury, or sulphur from sulphuric acid; but there were others that I could not identify with any mapped lines. The well-known variability in the spectra of some substances under varying conditions of degree of evacuation of the tube, led me to ascribe similar causes for these anomalous appearances, and to reject the suggestion made by one of us in a doubtfully serious spirit, that a new element might be in question.

Hillebrand made one observation, the significance of which at that time (prior to the discovery of radioactivity), was not appreciated; namely, that the amount of the supposed nitrogen in uraninite seems generally to bear a relation to the amount of contained uranium, or radioactive substance, in the mineral. As referred to later on in this paper, in general the amount of helium found in a mineral varies with the proportions of radioactive substances.

Hillebrand, who passed away a few years ago, was recognized as the master analyst wherever science is known, and methods developed by him are used throughout the world (19).

Lockyer has indicated that if the conditions under which Hillebrand made his spectroscopic examination of the gas he obtained had been different, he possibly would have seen the D_3 line, as Ramsay did, later. For Lockyer brings out the point that under certain electrical and pressure conditions the brilliant, fluted nitrogen spectrum in the yellow could flood that region and thus drown out the D_3 helium line, while under other conditions, for instance those surrounding the observations of Ramsay, the fluted spectrum in the yellow would be absent (4, pp. 278 and 284).

Also, Ramsay has stated (18) that --

> The characteristic spectrum of argon is almost completely masked by the presence of a few parts percent of nitrogen or of hydrogen; and that of helium is similarly affected, although to a less degree.

Though no quantitative experiments have been made on the subject, yet I should judge that the presence of from 5 to 10 percent of nitrogen entirely obscures the characteristic yellow line; the other strong lines still remain visible.

Hillebrand, himself, as appears in the above quotation from his letter to Ramsay, was aware that variations can be presented in the spectra of substances, but this apparently served to deter rather than to inspire further investigation by him, although the possibility "that a new element might be in question" was suggested at the time, as stated by him.

Professor William Ramsay and Lord Rayleigh Discover Argon

In the summer of 1894, William Ramsay, professor of chemistry at University College, London, in conjunction with Lord Rayleigh, discovered a new rare gas in the air (17, pp. 15-22). This was the outcome of previous researches conducted by Lord Rayleigh on the vapor density (comparative heaviness) of the principal gases, during which he came upon the epoch-making discovery that "nitrogen" separated from air was of greater density than that prepared from nitrogen compounds. Rayleigh and Ramsay proved that this was due to the presence of a new gas mixed with the nitrogen. On August 13, 1894, this gas was christened "argon", from the Greek word meaning "idle", on account of its inertness, by H. G. Madan (17, p. 22), then Chairman of the Chemical Section of the British Association.

Ramsay's Discovery of Helium in Cleveite

On the alert for other sources of argon, and hearing from the mineralogist, Sir Henry Miers (4, p. 284; 22; and 27, p. 235), of the British Museum, of Hillebrand's experience with uraninite, Ramsay instantly determined to repeat the latter's experiments and to carry them further on the chance of finding argon as a constituent of the gas obtained. With an ardor characteristic of the man, he hastened to secure a sample of the mineral cleveite (one of the uranium-bearing minerals), named for the Norwegian Professor Cleve of Upsala University, Sweden; powdered it; boiled it with weak sulphuric acid; sparked the gas thus obtained with oxygen over soda to remove the oxides of nitrogen thus formed; removed with pyrogallol the excess of oxygen added to the gas; washed the residual gas with boiled water; dried it, and examined it in a vacuum tube with the spectroscope. All this was carried out within 3 days of hearing of Hillebrand's work (21). The lines of argon were found, to be sure, but, to Ramsay's great astonishment, the brilliant yellow line characteristic of helium also appeared. This was in March 1895 (17; 20; 22). At Ramsay's request, Professor (later Sir) William Crookes (23) measured the wave length of the yellow line and confirmed its identity with the line found in the spectrum of the sun. The D_3 line has been found to be really a double line (24).

Thus was helium first recognized as a constituent of the earth's substance. Its discovery by Ramsay was announced in simultaneous communications to the British Royal Society and the French Academy of Sciences on March 26, 1895 (17; 20).

Other Discoveries of Helium

In the same year, Ramsay discovered helium in a specimen of meteoric iron from Augusta County, Va. (18), and Prof. H. Kayser of the University of Bonn, Germany, discovered helium, with the spectroscope, as a component of the atmosphere, at Bonn. The latter also found helium in the gas emitted from the mineral springs at Wildbad in the Black Forest (25). Helium was separated from the atmosphere by Ramsay and Travers in the summer of 1900 (17, pp. 122-125).

DISCOVERY OF OTHER RARE INERT GASES

With regard to the discovery of other members of the family of rare, inert gases, argon was detected through dual investigations by Lord Rayleigh and Sir William Ramsay, already referred to; krypton, neon, and xenon, in this order, were discovered in the atmosphere by Sir William Ramsay and Dr. M. W. Travers in 1898; and niton (so named by Whytlaw-Gray and Ramsay) (26), or "emanation", now also called "radon", was discovered by Prof. E. Rutherford in thorium compounds in 1900 (28). It is also emitted by radium and actinium. The emanations from thorium and actinium are very short-lived--a matter of seconds; that from radium (detected in 1900 by Dorn after Rutherford's discovery) is far more persistent. It loses one half its activity in about 4 days. The atomic weight of niton was established by Ramsay and Dr. Whytlaw-Gray in 1910 (26).

Rare Gases in the Atmosphere

Argon is contained in the ordinary atmosphere in the proportion of 1 volume in almost 107; neon, about 1 volume in 55,000; helium, about 1 in 185,000; krypton, 1 in 20,000,000, and xenon, 1 in 170,000,000 (29). Niton is contained in exceedingly small amounts in the air.

Prediction of Finding Rare Gases Before Discovery

The research work by which these gases were discovered in the atmosphere and in minerals was of the most brilliant character. The discovery of neon was predicted, with great accuracy, by Ramsay, before it was actually known, and the discovery of other gases was declared certain by him (17, chaps. VII, VIII, and IX). He drew his deductions from data relating to other elements then known, according to the so-called "Periodic Law", formulated independently by Mendeleef, of Russia, and Lothar Meyer, of Germany, which law predicates that the properties of the elements are periodic functions of their atomic weights. Consonant with the Periodic Law (now accepted as a fundamental law of chemistry), all of the chemical elements may be comprehended in a system in which they occupy fixed places and fall into characteristic groups according to, or as a function of, the weights of their respective atoms. Into some of the places for elements in this system which were vacant before the rare gases were known, it was found that they fitted, their properties being largely foretold by analogy even before their discovery.

Refinement of Technique Sometimes Required by Gas Research-- Niton

So superb was the research technique of Ramsay and Whytlaw-Gray, that in investigating and determining the properties of niton, in 1910 (26; 27, p. 284), they were able to make a series of experiments each of which was performed with the incredibly minute amount of about 1/10 cubic millimeter ($\frac{1}{164,000}$ cubic inch) of the gas, weighing about 2/1,000 milligram ($\frac{1}{32,400}$ grain), contained in a capillary tube. This was a bubble scarcely large enough to see with the naked eye, much less to manipulate. It was, however, sufficient for these masters. They used in this connection a so-called "microbalance" that was sensitive enough to weigh accurately 1/500,000 milligram ($\frac{1}{32,400,000}$ grain). This instrument was specially constructed by them, having a beam made of little quartz (silica) rods fused together, all enclosed in a brass case, from which the air could be exhausted. As a counterpoise on the balance a small silica bulb of but 22.2 cubic millimeters (about 1/739 cubic inch) capacity, containing air, was used, and the weighing was effected by regulating the air pressure in the brass case. Silica was chosen as the material for the balance beam, since it is very strong and its expansion by heat is neglible. To make the work still more difficult it had to be done with speed, for, in the space of less than 4 days, one half of the niton became decomposed.

In spite of the infinitesimal amount of niton available to them, Ramsay and Whytlaw-Gray managed to determine some of the physical properties of the gas, but it was necessary in this work to resort to the use of the microscope in making some of the observations (27, p. 290). This digression from the main theme of the narrative has been made to give some idea of the order of refinement which requirements in the field of gas investigation can sometimes demand.

PROPERTIES OF RARE GASES

As already mentioned in this paper, helium is one of a remarkable family of gaseous elements, the other members of which are neon, argon, krypton, xenon, and niton (known also as radon). Their extreme chemical inactivity, or inertness, sets them apart from all other substances. They are completely resistant to ordinary chemical change (30). Respective hydrates of argon, krypton, and xenon are reported to have been prepared, and certain investigators have been active in research work aimed to develop evidence of the possible formation in minute amounts of compounds of helium, argon, krypton, and xenon, with some other elements, under specifically directed and controlled electrical and subatomic stimulus. To date, however, these researches and the results attained are of purely scientific interest (31).

For all practical purposes these gaseous elements are nonreactive. This group, is, therefore, known as the "inert gases", or, by reason of their comparative scarcity, the "rare" or "noble" gases. Their aloofness characterizes

them as the aristocrats among the chemical elements. They are colorless, odorless, and tasteless, and since they are inert, are nonpoisonous in the ordinary acceptance of the term. They are noninflammable and nonexplosive. Helium, the first to be discovered (as an unknown element in the sun), is the lightest of the series. Next to hydrogen it is the lightest known substance. The weight of a normal liter of helium (a liter at a temperature of 0°C. and a pressure of 760 millimeters of mercury at 0°C.) is 0.1785 gram. The weights of normal liters of air and hydrogen are 1.2929 grams and 0.08988 gram, respectively (32). The specific gravity of helium (air being taken as 1) is therefore $\frac{0.1785}{1.2929}$, or 0.1381, while that of hydrogen is 0.06952. It will therefore be seen that air weighs about $7\frac{1}{4}$ times as much as helium and $14\frac{1}{2}$ times as much as hydrogen.

Lifting Power of Helium and Hydrogen

Questions have often been asked the Bureau of Mines as to the lifting power of helium. The lifting power of one gas in another depends upon the difference in their densities, or the difference in the weights of respective equal volumes under like conditions of temperature and pressure. For instance, at 60° F. and under the average pressure of air at sea level (14.70 pounds per square inch, which is equivalent to the pressure exerted by a column of mercury 760 millimeters, or 29.92 inches, high) 1,000 cubic feet of air weighs 76.36 pounds and the same volume of helium weighs 10.54 pounds. Under these conditions of temperature and pressure, 1,000 cubic feet of pure helium has a lift in air of 76.36 minus 10.54, or 65.82 pounds; that is, each cubic foot has a lift of 0.06582 pound. One thousand cubic feet of hydrogen, under these same conditions weighs 5.31 pounds, giving a lift in air of 76.36 minus 5.31, or 71.05 pounds--that is, 0.07105 pound for each cubic foot. Helium will, therefore, lift $\frac{65.82}{71.05}$ times 100, or 92.64 percent as much as hydrogen.

The helium, as it comes from the Bureau of Mines production plant at Amarillo, Tex., is better than 98 percent pure. The Government's dirigible, the Macon, holds 6,500,000 cubic feet of such helium, having a gross lifting power of a little over 200 short tons.

Neon is somewhat lighter than air, though slightly more than five times heavier than helium. All of the other members of this series are heavier than air, niton being the heavyweight of the family. It outweighs air about $7\frac{1}{2}$ times.

Why a Steel Cylinder Filled with Helium Does Not Rise in the Air

The question has several times been asked the Bureau why a standard-sized steel cylinder filled with compressed helium gas does not rise in the air, since helium is so wonderfully light. The idea in the minds of the interrogators seems to have been that the more of this light gas that can be forced into the cylinder, the more it will lift.

There are two main conditions that affect the density of a gas, or to state it more simply, the heaviness or the lightness of a gas. These are temperature and pressure. If a gas is heated, the pressure remaining constant, it expands; it becomes thinner and the same amount of the gas is thereby made to occupy a greater volume; it is therefore lighter per unit volume. If, on the other hand, the gas is cooled, under constant pressure, it contracts and there is consequently more of it occupying the same given volume; it becomes denser and consequently, heavier. Furthermore, if the pressure on a gas is increased, the temperature remaining constant, it is squeezed together, becomes denser and consequently heavier per unit of volume. If the pressure is decreased, the temperature remaining constant, the gas expands, becomes thinner and, consequently, a given volume of it is lighter. It will therefore be readily seen that the more helium there is compressed into a cylinder, the heavier the gas will become. Helium gas is transported and stored in standard-sized steel cylinders which are of about a foot and a half actual space capacity. The helium is compressed under a pressure that exerts a force of about 1,800 pounds on each square inch of the inside of the cylinder wall. When the helium in the cylinder, under these conditions, is released to ordinary atmospheric conditions of pressure and temperature, it will occupy a volume of about 178 cubic feet. So 178 cubic feet of helium has been compressed into the space of $1\frac{1}{2}$ cubic feet. In consequence of this it has been made so dense that it has now become about 17 times as heavy as the surrounding air. It will therefore not exert any lifting power, but, on the contrary, adds about 2 pounds to the 130 pounds, more or less, that the cylinder itself weighs.

Gradation of Properties in Rare Gases

Beginning with helium and running through neon, argon, krypton, xeon, and niton, in this order, there is to be observed a general gradation of certain physical properties of the remarkable family of rare gases to which helium belongs. This is noted with regard, for instance, to density, molecular weights (the weights of the respective free-existing, infinitesimally small physical particles of which they are composed), refractive index and dispersion (or the way light is changed on passing through them), and boiling and melting points (33; 31). In some properties, such as heat conduction, solubility in water, viscosity (resistance offered by a substance to the relative motion of its particles), and compressibility, the progression is broken to some extent for no reason now definitely known. All of these gases are monatomic; that is, their molecules contain but 1 atom. Therefore, their respective molecular weights are the same as their atomic weights. In the case of helium this weight is 4 (oxygen gas being taken as a standard, or 16). Helium is the most difficult of all gases to liquefy and solidify.

The Absolute Scale of Temperature

The theoretical point of absolute cold lies at -273.1°C. (-459.58°F.). This point is called the absolute zero. Temperatures reckoned from this zero are called absolute temperatures (Abs.), or Kelvin temperatures (K.) in honor of the distinguished English physicist, Lord Kelvin. At the absolute zero, molecular motion ceases entirely and there is no heat, since heat is but a

manifestation of molecular energy. In the absolute system of temperatures, either the centigrade or the Fahrenheit interval of degree may be used. In practically all scientific work, however, it is standard practice to determine temperature by the centigrade scale. In this scale zero temperature, 0°C., is the temperature of melting ice or freezing water, and 100°C. is the temperature of boiling water when the pressure, in each case, is 1 atmosphere (760 millimeters of mercury). In the Fahrenheit scale, 32°F. is the freezing temperature of water and 212°F. that of boiling water, at 1 atmosphere pressure, with an interval of 180° between, compared with 100° on the centigrade scale, or a ratio of 9 to 5

Gas Measurements--Laws of Boyle and Charles

The following elemental discussion of how gas measurements may be calculated according to the so-called "Gas Laws", is given for the information of any reader who may be specially interested.

The two laws evolved, respectively, in the seventeenth century by the great Irish scientist, Robert Boyle, who was the son of the Earl of Cork, and in the eighteenth century by J. A. C. Charles, the eminent French mathematician and physicist, are often used to compute changes of gas volume with changes of pressure and temperature. Charles, by the way, was the first to employ hydrogen in balloons. This was in 1783 (2a).

Boyle's Law

Boyle's law states that, if the temperature of a given mass of gas is kept constant, the volume varies inversely as the absolute pressure (the pressure starting with the true zero of pressure); or, the product of the volume times the absolute pressure remains constant. To illustrate the application of this law, consider a quantity of gas that occupies a volume of 100 cubic feet when its temperature is 0° C. and its absolute pressure is 1 atmosphere (14.7 pounds per square inch). If the absolute pressure is raised to 2 atmospheres (29.4 pounds per square inch, or twice the original pressure), and the temperature remains at 0° C., the volume will be reduced to one half of the original volume, or 50 cubic feet. If the absolute pressure is raised to 100 atmospheres (1,470 pounds per square inch) the volume will be reduced to $\frac{1}{100}$ the original volume, or 1 cubic foot.

In this connection, it should be stated that gages, by which pressures are ordinarily measured, do not register absolute pressure. Their zero point represents 1 atmosphere pressure. They therefore show the difference between the pressure within the container to which they are connected and the pressure of the atmosphere surrounding the gage. That is, if a pressure gage surrounded by air under atmospheric pressure at sea level (14.7 pounds per square inch) is connected to a vessel that contains gas under 2 atmospheres absolute (29.4 pounds per square inch) the gage will show only 1 atmosphere (14.7 pounds per square inch). If it is connected to a vessel that contains gas under 100 atmospheres absolute (1,470 pounds per square inch) it will show only 99 atmospheres (1,470 minus 14.7, or 1,455.3 pounds per square inch). Therefore, in order to obtain the absolute pressure of gas within a vessel it is necessary to add the pressure of the atmosphere to the pressure reading shown by a gage connected to the vessel.

Charles' Law

Charles' law states that if the pressure of a given mass of gas remains constant, the volume varies as the absolute temperature (the temperature scale whose zero is 273.1° C. below 0° C.); or, the volume divided by the absolute temperature remains constant. To illustrate the application of Charles' law, again consider the quantity of gas that occupies a volume of 100 cubic feet when its temperature is 0° C. (273.1°K.) and its pressure is 1 atmosphere. If the absolute temperature is raised to 273.1° C. (546.2° K.) and the pressure is not changed, the volume will be twice the original volume, or 200 cubic feet. The absolute temperature has been doubled; therefore, the volume has been doubled.

Laws of Boyle and Charles Combined

If the laws of Boyle and Charles are combined, we find that the volume of a given mass of gas, times its absolute pressure, divided by its absolute temperature, is a constant quantity. Expressed mathematically, using V to represent the volume, P the absolute pressure, T the absolute temperature, and C the constant quantity,

$$\frac{PV}{T} = C.$$

To illustrate the combined laws, suppose we take a quantity of gas having a volume of 100 cubic feet at 0° C. (273.1°K.) and 1 atmosphere of pressure. Assume that the absolute pressure is raised to 100 atmospheres and the absolute temperature is raised to 100°C. (373.1°K.) and it is desired to know the new volume of the gas under these conditions. Let V' represent the unknown new volume and P' and T' represent the new pressure and temperature, respectively.

According to the foregoing expression, since

$$\frac{P'V'}{T'} = C, \text{ and } \frac{PV}{T} = C,$$

then

$$\frac{P'V'}{T'} = \frac{PV}{T}.$$

To find the new volume which has become, under the new conditions, the unknown quantity, V', the known quantities may be substituted in the equation, giving

$$\frac{100 \times V'}{373.1} = \frac{1 \times 100}{273.1}$$

from which

$$V' = 1.3662 \text{ cubic feet.}$$

Strictly speaking, the laws of Boyle and Charles apply only to what may be conceived as the hypothetical "perfect" or "ideal" gas. Since the days of Boyle and Charles, it has been found that real gases do not follow these laws exactly; they may be more compressible or less compressible than the hypothetical "perfect gas", depending upon their composition and the initial and final temperatures and pressures involved in the change. However, for computation of changes of volume of gases, when they are well removed from temperatures and pressures under which they liquefy, these laws may be used without introducing large errors, and it is beyond the scope of this paper to develop the mathematical formulas to take into account departures from them.

Compressibility of Helium

Within ordinary ranges of temperature and pressure, helium is slightly less compressible than the "perfect" gas would be. For instance, it has been found experimentally that 100 cubic feet of helium at 0°C. and 1 atmosphere pressure would be reduced to 1.4167 cubic feet if its temperature were increased to 100°C. and its pressure were raised to 100 atmospheres. Since the volume for the ideal gas would be 1.3662 cubic feet, the volume of the helium would therefore be 3.7 percent greater. To that extent it is less compressible than the ideal gas under the specified conditions.

Further Properties of Rare Gases

Although helium does not conduct heat quite as well as hydrogen, which is the best heat conductor of all gases, it is about six times as good a conductor as air (34, vol. V., p. 213). It conducts electricity better than any other true gas, except neon. Collie and Ramsay (35) reported that a spark discharge 250 to 300 millimeters long could be obtained in helium under conditions as to electric potential and pressure of gas that produced spark discharges in other gases with mean results as follows: Oxygen, 23.0 millimeters; air, 33.0 millimeters; hydrogen, 39.0 millimeters; and argon, 45.5 millimeters.

Liquefaction of Helium

It is necessary to reach the extremely low temperature of about 5.2°C. above the absolute zero (5.2°K. = -267.9°C. = -450.2°F.) in order to liquefy helium (34, vol. III, p. 248). Helium was first liquefied by the Dutch scientist, Prof. Kamerlingh Onnes, at the University of Leiden, on July 10, 1908 (36), since when its liquefaction has been accomplished by Professor McLennan and associates (37) of the University of Toronto; by Walter Meissner, March 7, 1925, at the Physikalisch-Technische Reichsanstalt, Berlin (38); by Professor W. H. Keesom of Leiden (39); by Dickinson, Brickwedde, Cook and Scott (41) on April 3, 1931, at the U. S. Bureau of Standards, Department of Commerce, Washington, D. C.; by Goez and Focke at the California Institute of Technology (72); at Oxford University, England; at Breslau; and possibly elsewhere.

Liquid helium is colorless, mobile, and transparent. Next to liquid hydrogen, it is the lightest liquid known. Its density at its boiling point is 0.126, water being taken as a standard, or 1, while the density of liquid

hydrogen at its boiling point is 0.071 (34, vol. I, p. 102; vol. III, p. 20). The maximum density of liquid helium is at 2.3°K. (34, vol. I, p. 102). It boils under atmospheric pressure at about 4.2°K. (34, vol. III, p. 203), has a critical temperature (the temperature above which it cannot exist as a liquid at any pressure) of 5.2°K., and a critical pressure, or pressure at the critical temperature, of 2.26 atmospheres (34, vol. III, p. 248).

Onnes liquefied helium, prepared from the mineral monazite, by passing the gas through an apparatus in which it was cooled, first to the temperature of liquid air (about -192°C., or -312°F.) and then to 15°K. (about -432.6°F.), by surrounding it with liquid hydrogen boiling under reduced pressure and thereupon expanding the helium through a nozzle or valve. By this means, 60 cubic centimeters (about 2 fluid ounces) of liquid helium was obtained in 3 hours from 300 liters (about 10½ cubic feet) of the gas.

Solidification of Helium

Professor Keesom, another Dutch scientist, was the first to solidify helium, which he did at Leiden, June 25, 1926 (39). Onnes had tried to solidify the gas by lowering the temperature alone (39) and attained the extremely low pressure of less than 1/50 millimeter of mercury, at which the temperature was estimated to be 0.82°K., but the helium remained liquid.

It is interesting to record that Professor Giaugue and D. P. McDougall, on April 9, 1933, while carrying out experiments on demagnetization of gadolinium sulphate, starting with a temperature of 1.5°K., produced with liquid helium, reached a temperature of 0.25°K., just a quarter of a degree above the point of absolute absence of heat (40).

Professor Keesom succeeded Professor Onnes at the University of Leiden, on the untimely retirement and subsequent death of Onnes. Keesom used the cooling method of Onnes. In the final stage a surrounding medium of liquid helium itself, boiling under reduced pressure, was employed. However, he also applied pressure, which was the further requirement necessary for success. On July 1, 1926, continuing his experiments, he solidified helium at various temperatures from 4.21°K. down to 1.19°K., according to the pressure employed, from 140.5 atmospheres or 2,065 pounds per square inch to 25.3 atmospheres or 372 pounds. A homogenous, transparent, crystalline solid was obtained which was so tough that it could be hammered.

Simon, Ruhemann, and Edwards succeeded in solidifying helium at 20°K., and about 1,800 atmospheres pressure. This is remarkable, since the solidification temperature under this condition is about four times as great as the gas-liquid critical temperature (42).

Thus helium, the most difficult of all gases to liquefy and to solidify was finally conquered by the skill and perseverance of the scientist.

Laboratory Method for Separation of Helium from Other Gases

Helium can be separated from other gases, except hydrogen, by a process of adsorption. On passing the gas mixture through specially prepared, or activated, cocoanut charcoal at the temperature of liquid air, exercising careful control and, if necessary, repeating the process, the accompanying gases can be completely adsorbed and the helium, spectroscopically pure, can be pumped off. This method is used by the Bureau of Mines in determining the helium content of natural gas. No authentic instance of natural gas containing hydrogen has come to the Bureau's notice in its helium-bearing gas investigations.

SOURCES OF HELIUM

Radioactive Substances

After helium was discovered in minerals, it was found that this element is a product of the spontaneous disintegration of radioactive substances, as shown by Ramsay and Soddy (43). These strange everchanging bodies, three series of which--uranium-radium, thorium, and actinium--are known, are found in minerals and rocks (44; 62, p. 11). Their atoms give off, or emit, three types of radiant energy, called alpha, beta, and gamma rays. The alpha ray, or particle, emitted at an enormous rate of speed, of the order of thousands of miles per second, is a helium atom carrying two positive charges of electricity. In other words, the alpha ray is an evolved, electrically-charged helium atom. This loses its charges spontaneously and becomes finally ordinary helium.

Recent research experiments have shown that helium may be produced artificially by bombardment of certain substances (lithium, boron, some hydrocarbon compounds, etc.) with high-velocity electric particles--protons and alpha rays (45).

Radioactive minerals are very widely distributed throughout the earth's crust, but in comparatively small aggregations; consequently, helium also is found in many places in the earth's outer substance (46) but generally, excepting in certain natural gases to be mentioned later, in neglibible amounts.

Mineral Springs and Volcanic Gases

Helium is present in minute quantities in natural waters, and has been found in the gases given off by certain mineral springs, and in volcanic and fumarole gases. In the boracic acid fumaroles, or openings in the earth giving forth steam, fumes, and gases at Larderello, in Tuscany, Italy, helium is evolved in small content, of the order of 0.10 to 0.15 percent (47). In gases given off by hot springs in Iceland, helium was found only to the extent of 0.0006 to 0.0104 percent (48).

Origin of Atmospheric Helium

The presence of helium in air, already referred to, has been considered by some to originate in its evolution during the disintegrating changes in the radioactive minerals of the earth. Others account for it on the theory of primordial existence. At the earth's surface, the air contains about 1 volume of helium in 185,000 volumes. It has been asserted that the atmosphere at the height of 100 miles is principally hydrogen and helium, and at 500 miles in composed entirely of these two gases.

Minerals, First Source of Helium

Naturally, minerals were the first source of recovery of helium. The amounts obtainable were, however, limited, being sufficient only for scientific investigational work, while the cost of production was prohibitive for other than such purposes. It is estimated that, before helium was produced from natural gas, the cost of obtaining it from minerals amounted to as high as $2,500 per cubic foot. Probably not more than 100 cubic feet had been produced in the world, most of which precious hoard was owned by Professor Onnes of Leiden, Holland. Contrasted with this, as will be referred to later, in a single month, January 1932, more than a million and a half cubic feet were isolated in the production activities of the United States Bureau of Mines helium plant, near Amarillo, Tex., from natural gas of the Cliffside structure, of the Amarillo gas field, at a net operating cost for that month of about one half cent a cubic foot.

Radioactive Ores

Among the minerals, it is radioactive ores of the groups containing uranium and thorium in which, as a general rule, helium is to be looked for in appreciable amounts. The initially active substances in these minerals are the uranium and thorium. These original elements through spontaneous decomposition produce three long series of other radioactive substances, which themselves undergo further transmutations, one into the other. The final product arrived at, in each series, is lead. This lead appears to be the ultimate member of the series, as it exhibits no further radioactivity. At certain disintegration stages, in this evolution of new elements, atoms of helium are given off (44).

Helium Contained in Minerals

The amount of helium in the mineral depends in general on its amount of contained uranium or thorium, the age of the mineral, and its capacity for retaining or occluding this element. Strangely enough, however, there are some minerals rich in helium that contain little or none of these radioactive substances: the beryls, for instance. However, just recently (49) a new instance of radioactivity, the explosion of the beryllium atom, has been discovered. Measurements by Rutherford and Boltwood indicated that a gram of radium (disintegration product or uranium) will produce helium at a rate of

164 cubic millimeters per year. Cleveite, pitchblende, carnotite, fergusonite, samarskite, thorianite, and monazite may be mentioned as radioactive mineral sources of helium. The last-named, being found in greater quantities, offers the best raw material of this character. Monazite sand, a mineral composed of phosphates of cerium and other rare earth metals and containing 5 to 10 percent thoria (thorium oxide), is found in comparatively small amounts, of no present commercial importance, in the United States (50). The largest supplies are located in southern Brazil, and in the province of Travancore, on the southernmost tip of India. Monazite is exported from India in rather large tonnage for its content of thoria, from which incandescent gas mantles are made.

Recovery from Minerals

The helium may be obtained from such minerals by simply heating the finely ground substance to suitable temperatures; also by heating with dilute sulphuric acid, in case the mineral is soluble in that reagent; otherwise by fusion with acid potassium sulphate. The helium given off must then be purified to remove contaminating gases, such as nitrogen, argon, and in some cases hydrogen, carbon dioxide, carbon monoxide, and hydrocarbons. About a quarter of the contained helium may be liberated simply by pulverizing the mineral to a sufficient degree of fineness and pumping off the gas, as the helium is held merely occluded, as such, in minute cavities or pores in the mineral, and is not in actual combination. Helium has been recovered from cleveite at a rate of 7.2 cubic centimeters per gram; from thorianite, 3.9 cubic centimeters; from monazite sand, 2.41 cubic centimeters.

Separation of helium from monazite sand has been carried out in the chemical research laboratory at Teddington, Middlesex, England (51). A recovery of between 26 and 27 cubic feet of 96.5 percent helium per short ton of sand was effected by heating the material to high temperatures and purifying the gases liberated. By purifying with metallic magnesium and calcium, heated to red heat, a purity of at least 99.5 percent was attained. The sand was imported from Travancore, India, and contained helium of the order of 1 cubic centimeter per gram.

Helium in Gases of Mineral Springs and Mines

Helium has likewise been collected from gases given off from mineral springs, but only in limited amounts. Although the percentage of helium content of some of these gases, in Europe, runs comparatively high--over 10 percent in at least one instance--the total amounts of the gases themselves do not permit of consideration for any but limited helium recovery. It has been estimated that the Cesar Spring at Neris (Allier), France, one of the largest of such sources of helium, evolves about 1,200 cubic feet of the gas per year. The gases from these springs consist in a few instances chiefly of carbon dioxide; the great majority are rich in nitrogen, resembling in this respect some of the natural helium-bearing gases of the United States (52; 62, pp. 49-51). A number of examinations by the Bureau of Mines of gases in the United States from springs, both hot and cold, and of geysers have

failed to reveal more than traces of helium. The Bureau has also examined oil shale, coal, and gases from coal and metal mines without finding any appreciable amounts of helium. Helium has been found in small percentages in gases of French and Belgian coal mines (62).

Discovery of Helium in Natural Gas from Dexter, Kans., by Cady and McFarland

In 1903 a rather shallow gas well which, however, gave a strong flow of gas was drilled at Dexter, Cowley County, Kans. In consequence there was great rejoicing in the little village. The advent of this gas was hailed as the harbinger of great prosperity. Visions of Dexter expanded to the proportions of an important center of natural gas activity and wealth swam before the eye. But the joy was premature and short-lived; the goose that was to lay the golden egg died before the egg appeared, for, to the consternation of all concerned, the gas was found to be almost noninflammable and practically worthless as fuel. It would not burn unless played into a fire kept going by ordinary combustibles. A sample of the gas was therefore sent to the University of Kansas for examination, in order to discover the cause of its astonishing, perverse behavior. There, on analysis, it was found (53, 54) that the gas contained very little combustible matter, not over 15 percent, but was over 80 percent nitrogen and other inerts. In view of the work of Ramsay and Rayleigh in connection with the discovery of argon, this "nitrogen" was subjected to some sparking with oxygen, but the residuum was reserved for further analysis when time should permit. Later, in 1905, Professor H. P. Cady and Dr. McFarland subjected this residuum to carefully conducted further examination and found that it contained helium (55) and that the gas from the Dexter well had a helium content of 1.84 percent. Thus was helium for the first time discovered as a constituent of natural hydrocarbon gas.

Helium Discovered in Other Natural Gases

Following the discovery of helium in the gas from Dexter, Cady and McFarland, pursuing their investigations further, succeeded in finding helium in a large number of gases from Kansas and a few from other States (55, 56). They also found argon and neon in some of these gases.

A gas from Augusta, Kans., that contained 1.97 percent of helium was analyzed by C. W. Seibel (57) for neon, argon, krypton, and xenon. He found that all of these gases were present, and he showed that their relative proportions in the gas were different from those in which they are contained in atmospheric air.

Noncombustible Natural Gases Do Not Necessarily Contain Helium

From time to time the Bureau of Mines receives communications reporting the finding of gas issuing from shallow water wells, openings and fissures in the ground, and even from the surface, with the statement that the gas will not burn, and that it must, therefore, be helium. The correspondents are led to this conclusion because they have heard that helium is noninflammable.

However, helium has never yet been found naturally evolved in such a state of purity that it would extinguish flame, and the content of helium in even the richest of helium-bearing natural gases is so small as to have little effect on the inflammability or explosibility of the gas.

Although certain natural gases will not burn, it is the presence of unusually large amounts of constituents other than helium, such as nitrogen, as in the Dexter gas, or carbon dioxide, that prevents combustion. Nitrogen is present in the general run of ordinary natural gases to the extent of anywhere from 1 percent up to 30 or 40 percent, but it has been found in some cases as high as 99.66 percent. Carbon dioxide is ordinarily present in amounts of less than 1 percent, but some gases issuing from wells have been found to be almost pure carbon dioxide. As both nitrogen and carbon dioxide are noncombustible, the effect of their presence on the burning power of natural gases in amounts as above set out, will be obvious. Therefore, the fact that a naturally occurring gas will not burn is not necessarily an indication that it contains helium. In fact, the natural gas of the Petrolia gas field, Clay County, Tex., which was for some years processed for helium production by the Government, had satisfactory burning qualities, and the gas near Amarillo which is being used to supply the present nearby Government plant has excellent heating value. After passing through the helium-removal process the gas is used for fuel both in the plant and in the city of Amarillo.

Various Constituents in Helium-Bearing Natural Gas

Most helium-bearing natural gases contain some nitrogen, but, on the other hand, many of the gases that have a large nitrogen content do not contain enough helium to be detectable by ordinary methods of gas analysis. Oxygen is rarely present, and then only in small amounts. Such gases do not contain carbon monoxide or hydrogen, except possibly in rare instances, but some of them contain poisonous hydrogen sulphide, at times in concentrations fatal to animal life. A content of this lethal gas as high as 20 percent has been found.

Determining Heating Value of a Fuel Gas

The ordinary practical standard of measurement used to determine the heating value of fuels is the British thermal unit (B.t.u.), which is the amount of heat necessary to raise the temperature of 1 pound of water 1°F. The heating power of fuel gases is usually figured in terms of the number of B.t.u.'s produced by the combustion of 1 cubic foot of the gas under some selected standard condition of pressure and temperature. A number of such standards are in use. In natural gases it is the hydrocarbon content that furnishes the heat of combustion.

Helium-Bearing Natural Gases

As has been stated, helium in sufficient quantity for large-scale production and use is derived only from certain natural gases which contain the helium as such. Helium production from natural gas means simply the separation, or isolation, of the helium from the other gaseous constituents. The

rich helium-bearing natural gases are found in regularly drilled gas wells, extending to so-called "sands", or porous rocklike subterranean formations in which the gas is held captive. Neither natural gas nor petroleum is found in the earth in large, free, unbroken volumes, but is contained in the interstices of "sands" under pressure, called "rock pressure." Some of these helium-bearing gas sands are located comparatively near the surface, as, for instance, the original discovery well at Dexter of which the depth is only 325 feet; but generally they lie much deeper, as far down as 3,600 feet at least, or something more than three fifths mile, as on the Cliffside structure of the Amarillo gas field.

It has been found that natural gases containing helium are widely distributed over the United States, but so far only a few localities have been discovered in which the volume of the gases themselves and their helium content are sufficiently large for economically practicable helium production. These favorable conditions prevail in the region of northwestern Texas. However, in addition to helium content and volume of gas, economic production of helium depends on many engineering and other factors, such as efficiency and costs of process, cost of the gas (initial purchase cost, leases, royalties, etc.), marketing or disposal of residue gas after processing, topographical and geographical situation of the gas, available water and fuel supply for the production plant, transportation facilities and cost, distance to market, consumption demands, and market price. In some parts of Kansas, gases running as high as 3.5 percent in helium content have been found, but investigations carried out by the Bureau of Mines, with the cooperation of the United States Geological Survey, have so far indicated that the resources of the helium-bearing gases, themselves, in Kansas are less favorable than those in Texas. As far as is known, the same may be said of Colorado and Utah where gas running as high as 7 or 8 percent in helium has been discovered.

Bureau of Mines Helium-Bearing-Gas Survey

The Bureau of Mines has maintained for several years—in fact, ever since the entrance of the United States into the World War—a comprehensive survey of the natural gases and gas-producing areas of the United States, for helium for Government purposes. During this period several thousand samples of gas from wells located in the majority of the States have been examined. This investigation is still in active progress and forms a vitally important part of the Bureau's helium work, for it is obvious that the operation of the Government's lighter-than-air ships depends, in first instance, upon the maintenance of an adequate and continuing supply of helium-bearing gas. Incident to this survey the Bureau found that a gas in a dome on lands of the public domain near Woodside, Utah, contained about 1.3 percent of helium. This gas has no fuel value, as it has over 90 percent inert carbon dioxide and nitrogen, and the location of the dome, for several reasons of economic and engineering importance, is not particularly favorable for low-cost helium production; however, an area of about 12,000 acres, covering the dome, and insuring the gas against wastage from any gas operations which might be conducted in that locality, was set aside, as authorized by Congress, as a reserve resource for the Government's use in case of war-time emergency.

World's Helium-Bearing Gas Resources

The greatest known resources of natural gas in the world are in North America. These include all sorts of natural gas. So far, the United States is the only country in the world in which helium-bearing gases in amounts sufficient for helium production for aeronautical purposes have been found.

Canada

Helium occurs in the natural gases of several provinces of Canada, notably in Alberta and Ontario, but the amount of the helium-bearing gases is small. According to Elworthy (58), the richest of these gases, produced in Peel County, Ontario, has a helium content up to 0.82 percent, and Rosewarne and Offord (59) report gas from one well in the same province having 1.03 + percent of helium. The generally prevailing low helium content of Canadian gas and the consequent high cost that would be involved in processing it, do not permit of production on a scale large enough and at costs low enough for practicable use in aeronautics. Elworthy estimates that around 5,000,000 cubic feet of helium per year might be recoverable, but this assumes that most of the helium is to be extracted from gases containing as little as from 0.2 to 0.37 percent, with an 80 percent recovery, and that questions of cost are not to be considered. Experience in the United States has demonstrated that, in the present status of the art, production of helium from gas with less than 1 percent of helium is not attractive from an economic standpoint.

Europe and the Orient

Helium has been found in natural gases in some countries in Europe and the Orient, but only in extremely small amounts. Such countries are Australia and New Zealand, Japan, Transylvania (largest European natural gas-producing territory), and Germany. In the last-named country, helium has been just recently reported in Neuengamme, Heide, Oberg, Volkenroda, Leopoldshall, Ascheberg, and Ablen, but in no case was the occurrence as great as 0.2 percent.

PRODUCTION OF HELIUM

Historic Evolution

The production of helium on a large scale was born of the necessity of war, projecting the use of helium in military and naval aeronautics. Employment of lighter-than-air craft, except for specialized scouting activities, particularly in connection with strategic operations of the fleet at sea, was an exceedingly hazardous undertaking in the World War, due to the extreme inflammability of the hydrogen gas with which the craft were inflated. The hydrogen was easily ignited by incendiary bullets, and during the hostilities many of the German Zeppelins that were sent over England for bombing purposes were thus brought down in masses of flame. On the other hand, it has been reported that the German fleet never put to sea without one or more Zeppelin scouts flying far overhead, able, from the lofty vantage point of their far-flung field of observation, to sight enemy ships and give timely notice of their positions and movements.

Efforts of Ramsay, Threlfall, McLennan, and Others

As already described, the combined properties of inertness and buoyance of helium gas had been established by Ramsay and others long before the World War. These properties were recognized as ideal for a balloon gas, and scientists had considered the possibility of the use of helium in airships. The question was where to obtain the helium. Ramsay had examined mine gases in England without avail. He also knew of helium content in Canadian natural gas. But the first to give any practical impetus to solving the problem of acquiring helium on a large scale was Sir Richard Threlfall, in England, in 1915 (58; 60). Threlfall (who passed away on July 10, 1932) was aware of the helium content of natural gases in America. He was chairman of the Fuel Research Board, a member of the Advisory Council of Scientific and Industrial Research, and of other kindred organizations in England.

Search for Helium in Canada

Through representations that Threlfall made to the British Admiralty, a sum of money was appropriated and made available to Prof. J. C. McLennan, of the University of Toronto, to institute a search for helium within the British Empire (60) and, in the event that adequate supplies were found, to carry on experiments for the development of a process for its recovery. As a result, early in 1916, samples of gas from most of the producing Canadian gas fields were examined (58; 60), and it appeared, as an outcome of this work, that establishment of experimental helium production plants in Canada might be feasible at only two locations: namely, Hamilton, Province of Ontario, and Calgary, in Alberta. At no other place in the British Empire were helium resources found of any considerable importance.

Experimental Plant in Canada

In 1918, a small experimental plant was constructed and tried out near Hamilton, the work being participated in by Profs. John Satterly, E. F. Burton, and H. F. Dawes, and by John Patterson, Lang, and others, but the gas proved inadequate in amount and the plant was later moved to Calgary, to process gas from the Bow Island field. Finally, from December 1919 to April 1920 a series of trial runs was made. It has been reported that, in all, only about 60,000 cubic feet of 60 to 90 percent helium was produced before the plant was finally shut down.

Inception of Helium Production in the United States

In April 1917, shortly after the United States had entered the war, the American Chemical Society held its annual meeting at Kansas City, Mo., and on this occasion the paper by C. W. Seibel, already referred to (57), concerning the recovery of rare gases from natural gas, was presented. The reading of this paper gave rise to a discussion during which the late Dr. R. B. Moore, then of the Bureau of Mines, brought up the possibility of separating helium from natural gas in amounts practicable for lighter-than-air military service.

It was also in April 1917 that the Director of the Bureau of Mines placed G. A. Burrell (later Colonel Burrell) in charge of the Bureau's war-gas work. The Bureau of Mines had initiated this work, chiefly on gas masks, smoke screens, and toxic gases, in cooperation with the Army and the Navy. Burrell, who had been engaged in gas investigations for the Bureau of Mines for several years before the war, had analyzed samples of natural gas from many localities and had found helium in them. He had thought of helium as a possible filler for balloons and dirigibles.

Burrell thought of the possibilities of the gases of the Petrolia field, in Texas, in regard to helium yield on account of their large nitrogen content. He had samples sent to Dr. Cady for analysis. These samples were found to contain about 1 percent of the element. Thereupon, Burrell took up the question of utilizing this gas for helium recovery with the Director, with Dr. F. G. Cottrell, and others in the Bureau of Mines.

Experimental Production by the Bureau of Mines with Army and Navy Funds

The proposition to provide means for investigating the feasibility of large-scale helium separation from natural gas for military and naval aeronautics was initiated by Burrell, Dr. Moore, and other officials of the Bureau of Mines with the hearty approval of the Director. The matter was taken up by Burrell with the Army in a letter on May 12, 1917, to Major C. de F. Chandler of the Signal Corps. These activities resulted, eventually, in an allotment of funds from Army and Navy appropriations to undertake an experimental helium production program, the Bureau of Mines being authorized to conduct the technical work (61).

Pioneering Gas Investigations

From examinations of natural gas obtained from various localities in the United States, as well as geologic investigations by the United States Geological Survey (62) and the Bureau of Mines following the extensive pioneering work by Professor Cady and Dr. McFarland already mentioned (55 and 56), it was concluded that the Petrolia gas field, in Clay County, Tex., on the north-central border of that State, was the best source of helium-bearing gas at that time available, having the combined prime essential qualifications of adequate helium content and volume of gas.

Development of Production Process

The difficulties in the way of extracting one gas from another, when the desired gas is present in the comparatively insignificant proportion of less than 1 part in 100 parts, were at once apparent. This would mean undertaking the processing, in billions of cubic feet, of a raw material having over 99 percent undesired matter. However, since it was known that the natural-gas constituents other than helium could be liquefied at temperatures which would leave the helium, most difficult of all gases to liquefy, in a state of gas,

the employment of a method of separation based upon this circumstance was obvious. The principal constituents of the natural gas, methane, ethane, and nitrogen, could be separated from the helium in this way. Under the pressure of 1 atmosphere, ethane becomes liquid at -88.6°C. (-127.5°F.) (34, vol. III, p. 217); methane at -161.4°C. (-258.5°F.) (34, vol. III, p. 216), and nitrogen at -195.8°C. (-320.5°F.) (34, vol. I, p. 102). At the pressures usually obtaining in production cycles the liquefaction points of these gases are somewhat higher. Helium will not liquefy above a temperature of -267.9°C. (-450.2°F.) (34, vol. III, p. 248).

Cooperating Agencies--First Helium Produced

Since the process contemplated for helium production was analogous to cycles already in use for production of oxygen from the air by liquefaction, whereby the oxygen (about one fifth part of the air) is separated from the nitrogen (about four fifths part) by fractional distillation, several firms interested in that line of work were invited to cooperate with the Government. Three of these responded, and during 1918 three plants for experimental tryout of their respective oxygen-production processes, appropriately modified, were erected under cognizance of the Bureau of Mines (61). The firms cooperating were the Linde Air Products Co., the Air Reduction Co., and the Jefferies-Norton Corporation. One of these try-out plants was placed at Petrolia, Clay County, Tex., near the Petrolia gas field, and the other two were at Fort Worth, Tex. About 200,000 cubic feet of helium, all told, were produced by them in their experimental runs. About 147,000 cubic feet of this helium, compressed in steel cylinders, was on the dock at New Orleans, ready for shipment to France for use in observation balloons, when the Armistice closing the war was signed. In the operation of the experimental plants many technical data essential to economic, large-scale helium production were developed, and invaluable practical experience was gained.

Experimental Plants Point Way for Large-Scale Production, Then Pass out of Existence

Having served their pioneering purpose, the experimental plants passed out of existence after the close of the war, and a large production plant was then constructed in which was installed the Linde Air Products Co.'s modified production cycle.

Production Plant at Fort Worth

This was the U. S. Helium Production Plant, located just north of the city limits of Fort Worth, Tex. (63). It was constructed under cognizance of the Navy, with the Linde Air Products Co. cooperating, during the period May 1919 to April 1921. The separation equipment was operated under contract by the Linde Co., subject to the direction of the Navy, until July 1, 1925, when, pursuant to Congressional enactment, responsibility for the Government's entire helium conservation and production project, including the plant and

pipe line, passed under the jurisdiction of the Bureau of Mines (64). The plant was placed in an inoperative status, January 10, 1929, by reason of the approaching exhaustion of the Petrolia gas field, its source of supply.

In this plant was produced about 46,000,000 cubic feet of helium, figured on a 100 percent basis. The process represented no easy task. It involved many complicated scientific problems, for the working out of which but scanty technical data existed. For every cubic foot of helium recovered, over 120 cubic feet of gas had to be processed. The gas came to the plant from the Petrolia field, about 100 miles distant, through a Government-owned 10-inch pipe line.

Petrolia Gas Field

At the time of the erection of the production plant at Fort Worth, it was known that the Petrolia gas field would not last for many years, since the field was 10 years old when the Government's helium project started. However, it was then the only known source of helium-bearing gas sufficient in quantity and of high enough helium content to warrant consideration for large-scale helium production.

Nocona and Amarillo Fields

The foreshadowed exhaustion of the Petrolia gas field made it necessary for the Bureau of Mines to locate a new supply of helium-bearing gas. Two sources of such gas were discovered, one in Montague County, Tex., known as the Nocona gas field, and the other in Potter County, Tex., known as the Cliffside structure, in the Amarillo gas field. Although the Cliffside structure was the best of the known sources for large-scale, continuous, and prolonged production of high-grade, helium-bearing natural gas, the Nocona gas field was under serious consideration for a time, because it could be connected to the plant at Fort Worth. However, rapid oil development and consequent wastage of the gas by oil operators destroyed the Nocona field's desirability, and a decision was made to build a new plant near Amarillo.

The Government-Controlled Cliffside Structure, Source of Supply for Amarillo Project

The gas of the Cliffside structure is approximately twice as rich in helium as that from the Petrolia field. Governmental control has been so far secured as to assure that this gas can be conserved. This structure is located

a few miles to the north and west of the city of Amarillo, a modern progressive community of about 45,000 inhabitants, situated at about the center of the Texas Panhandle. Work toward the acquirement of gas reserves, anticipating erection of a plant at Amarillo, was started by the Bureau of Mines in 1926, when it was apparent that the supply of gas for the Fort Worth plant was rapidly failing. The Cliffside structure was then known to produce gas averaging about 1.75 percent helium content. In 1927, an operating contract for gas, with option (to purchase gas rights), covering 20,000 acres of land on the Cliffside structure was entered into by the Government. This was followed by exercising the option, and effecting other contracts and purchases. These together give the Government control of gas in about 50,000 acres of land. The reserve of gas in the structure and the Government's control over it assure the Army and Navy adequate supplies of helium for many years.

Drilling for gas to supply the Amarillo plant was started in February 1928, and upon completion of this work the Bureau of Mines was in possession of four helium-bearing gas wells, with a total open-flow capacity of about 30,000,000 cubic feet of gas per day, located on the Cliffside structure.

The Amarillo Helium Production Plant

In July 1928, construction of the pipe line from the Cliffside structure to the plant site was started and a contract was awarded for erection of the plant buildings, ground being broken early in August. The first separation unit of the plant was placed in regular production in April 1929, after helium in preliminary, or tune-up, operations had been recovered as early as in January. Since that time the efficiency of the plant was steadily increased and there has been a marked reduction in operating costs. A second separation unit of the plant was completed in May 1930. Both of these units were designed by Bureau of Mines technical men and were erected under their supervision. The separation apparatus of the second unit was built in the well-equipped machine shop maintained as an essential plant facility.

Description of Plant

This plant is known as the United States Bureau of Mines Amarillo Helium Plant (65). It is at Soncy, a point on the Rock Island Railway about 7 miles west of Amarillo, occupying a site of about 18½ acres, inclosed in a high, strong, wire-mesh fence. Highway No. 66 passes east-west, along the front of the plant, and the railroad parallels it outside the rear line of the reservation. The plant consists of an aggregation of eight major buildings of permanent construction, together with their appurtenances, including roadways; water-cooling pond; water wells and tower; gas holders; water, steam, gas, electric, and waste lines. A spur, or switch, leads into the plant enclosure from the main track of the railroad. Economic, engineering, and sanitary conditions for helium production activities at Soncy are very favorable. The altitude of the place is nearly 4,000 feet above sea level and climatic conditions are good. The water supply is adequate and good in quality, and the B.t.u. value of the gas is finely adapted to power production. The plant is connected with the wells on the Cliffside structure by a Government-owned, 6-inch, welded steel

pipe line, about 12 miles long, together with appropriate gathering lines, which operate at a pressure of from 600 to 700 pounds per square inch. The residue or processed gas is conducted from the plant to the City of Amarillo for industrial use, through a pipe line of 10-inch diameter.

Capacity of Amarillo Plant--Production and Costs

The plant at Soncy is capable, with its present two-unit equipment, of producing regularly and easily well over 24,000,000 cubic feet of helium per year. It has, however, never been run at its maximum capacity, because helium at that rate has not yet been demanded by the Government's military services. From the beginning of operations in April 1929 to May 1, 1933, the plant produced about 51,000,000 cubic feet of helium, figured on a 100 percent basis, at an average net cost for operation and maintenance of $9.60 per thousand cubic feet. Net cost, here, means expenditures in operating and maintaining the plant and gas field, less return from sale of residue gas. This cost, and those referred to in following paragraphs, are based on the actual helium contained in the airship gas supplied from the plant to the Army and Navy.

The unit cost of output depends to a large degree upon the rate at which the plant is operated. Items of cost which make up a considerable part of the total operating expense remain practically the same whether the output is large or small. Therefore, when the production is increased, there is only a slight rise in total operating expense, and consequently, a marked decrease in cost per thousand cubic feet of helium produced. Through experience gained in operation and economies and efficiencies introduced as time went on, the net average production cost for the fiscal year ending June 30, 1932, was brought down to $7.10 per thousand cubic feet.

As an illustration of what is possible with large output and under favorable conditions, special reference may be made to the month of January 1932. The costs for that month were the lowest thus far attained. Using only one of the two separation units, the plant produced 1,652,200 cubic feet of helium with expenditures of $12,544.48 ($7.59 per thousand cubic feet of helium produced) for operation and maintenance of the plant and gas field. There was a return of $4,401.50 from residue natural gas sold during the month, giving a net operating cost of $8,142.98, or $4.93 per thouand cubic feet of helium. This is less than one fifth of the lowest monthly net operating cost attained in the old production plant at Fort Worth.

Purity of Helium Produced

In addition to producing helium at greatly lowered costs, the Amarillo plant has improved the quality of the helium as it is produced and supplied to the Army and Navy. Prior to the operation of this plant, the average purity of the helium received by the military services was 95 percent, or less. Now, the Amarillo plant is regularly producing helium of better than 98 percent purity. Since an increase of even 1 percent in purity of the helium adds about 2 tons to the lift of a ship the size of the Macon, the increased purity of the Amarillo product is of great advantage in operation of the Nation's lighter-than-air craft.

In experimental work, looking to still further improvement in the quality of the plant's output, the cryogenic laboratory of the Bureau has succeeded in bringing the purity of a considerable quantity of helium up to 99.96 percent.

Helium versus Hydrogen as Balloon Gas

In the final analysis it costs less, under present conditions, to operate Government airships with noninflammable helium than it would to operate them with inflammable hydrogen. The first cost of hydrogen, if produced for airship operation, might be less than the present cost of helium; however, in the operation of lighter-than-air ships, there is a constant diffusion of air into the gas cells and of buoyant gas from the cells out into the surrounding atmosphere. This gradually lowers the lifting power of the gas and makes it necessary either to reinflate the ship with entirely new gas, or to purify the gas charge and add new gas to make up the lost volume. This has to be done several times a year.

Contaminated hydrogen must all be thrown away, for there is no known process for safe and economic purification of hydrogen in such large volumes as are required in aeronautics. On the other hand, helium may be drawn from the airship and purified at a cost of only 50 cents to $1.50 per thousand cubic feet, in plants that have been designed and built by the Bureau of Mines for the Army and Navy. The purified helium is then replaced in the ship and new, make-up helium added to restore the ship to its proper buoyancy.

Purification of Air-contaminated Helium

In purifying the helium, the contaminating air is liquefied by a process similar to that used in producing helium from natural gas. The impure helium is pumped under high pressure through a coil or purifier to which is attached a drain pot to catch and remove the liquefied impurities (air), the whole being surrounded by liquid air obtained in the operation of a separate cycle. This process cannot be applied to purify hydrogen, since the oxygen of the contaminating air might form with the hydrogen an exceedingly explosive mixture, which would probably blow up the plant. The helium required for replacement of gas losses in an airship over the course of a year's operation is only about 1 to $1\frac{1}{2}$ times the volume of the ship, compared to 8 to 10 times the volume in the case of hydrogen.

Helium Production Process at Amarillo

Most gases under pressure, when permitted to expand, cool themselves down. They also cool themselves when, on expanding, they are made to perform external work, for instance in the cylinder of an expansion engine. Both of these circumstances are made use of in the production of helium by liquefaction of the constituents of helium-bearing natural gas other than the helium contained therein.

Preliminary Removal of CO_2, and Other Contaminants From the Gas

The gas from which the helium is extracted at Amarillo contains carbon dioxide up to about 0.7 percent. This must be eliminated before the liquefaction process can be carried out, otherwise it would solidify at the low temperatures attained in the plant and plug the apparatus. It is removed, as a preliminary step, by scrubbing the gas with a 7 percent solution of caustic soda in an extended system of steel tubing under the pressure of 600 pounds at which the gas comes to the plant from the field. The water vapor accumulated in the scrubbing process, and some higher hydrocarbons of the gasoline series, are eliminated in the first part of a system of heat interchangers (or exchangers) through which the gas, still at 600 pounds pressure, now passes.

Production of Crude Helium

Through these interchangers the raw gas coming into the plant runs countercurrent with and is precooled by cold residual gas leaving the system, after having the helium removed. The hydrocarbon constituents of the gas and most of the nitrogen are liquefied and separated in this step. They are finally volatilized, pass out of the plant as residue gas, and are conducted through a pipe line to Amarillo for use as fuel. The gas phase remaining, consisting of about equal parts of helium and nitrogen, is collected at lowered pressure as "crude helium."

Crude Helium Stepped up to Better than 98 Percent Purity

This crude helium is next compressed and passes through a purifying process in which it is conducted through a second series of heat interchangers and finally enters a container (or pot) surrounded by liquid nitrogen, liquefied by the aid of expander engines in an auxiliary system. In this container, at the low temperature of about 312° below 0° F., and the high pressure of 2,500 pounds, the nitrogen of the crude helium is almost entirely liquefied and drawn off, while the helium, now at better than 98 percent purity, is reduced to ordinary temperature, through interchangers, and conducted to tank cars, or small steel cylinders, for transportation; or it is placed in a high-pressure storage system at the plant. A small part of the nitrogen is recovered in the purification process and used as a make-up for losses in the auxiliary nitrogen liquefaction cycle.

Within the Cycle

At the minimum temperature of about 312° below 0° F., reached in the plant, atmospheric air is a liquid, carbon dioxide and mercury are solids, lead becomes tough and elastic, and copper takes on a strength approaching that of steel. Rubber becomes as brittle as glass. Placing an icicle in the ultra-cold liquefied gases in this plant would be like thrusting a hot poker into tap water. The temperatures in the processing apparatus are measured by so-called "thermocouples": delicate instruments recording the temperatures in

terms of electric currents which are transposed into thermometric degrees. Each cubic foot of natural gas comes to the plant at ordinary temperature and under a pressure of about 600 pounds per square inch; it passes through the plant at express-train speed, is dropped in temperature to about 250° below 0°F., and is again raised to ordinary temperature, all in the sensationally small space of less than a minute of time. It leaves the plant at about 75 pounds pressure.

Transportation of Helium--Tank Cars and Cylinders

The tank cars in which helium is transported from the plant to the Army and Navy flying fields each consist of three or more large, heavy-walled, seamless steel cylinders mounted on railroad trucks. The Navy has at present several of these cars. The Army has two. They are specially designed and constructed for helium transport and accommodate helium at a pressure of from 2,000 to 2,250 pounds per square inch, which, when expanded to ordinary atmospheric conditions, coccupies a volume of from 200,000 to 215,000 cubic feet. By use of these cars a material saving is effected in transporting helium, on the score of reduced freight rates, saving of time and labor in handling, and a minimum of escape of gas on filling and discharging, as well as reduced leakage through valves.

The small cylinders, which are usually shipped in box cars, are of the standard hydrogen or oxygen type, each having an actual space capacity of about 1½ cubic feet and are shipped under a pressure of 1,800 pounds per square inch, holding an amount of gas which, when expanded to ordinary atmospheric conditions, occupies about 178 cubic feet. They are fitted with double-seated valves instead of the single-seated variety usually employed in handling other gases. Since it requires between eleven and twelve hundred of these cylinders, occupying two freight cars, to transport the same amount of helium as one tank car, the economies in the use of the tank cars will be obvious.

Helium Storage at Amarillo Plant

At the Amarillo plant, a permanent high-pressure storage equipment has been installed, consisting of about 24,000 small helium cylinders of the standard description mentioned. Their valves have been removed and the cylinders have been connected with a manifolded system of tubing. This storage has a capacity of about 3,500,000 cubic feet of helium when filled to a pressure of about 1,500 pounds per square inch.

Helium and Other Rare Gases Separated from Air

Comparatively small amounts of helium, as well as commercial volumes of argon and neon, are recovered as by-products in the industry of producing oxygen by liquefaction and separation of air. Helium and neon, which liquefy at temperatures far below those required to liquefy nitrogen, argon, and oxygen, may be drawn off as a gaseous mixture when the other constitutents of the air have been liquefied. The helium and neon may be separated by differential adsorption when

passed through activated charcoal at very low temperature. They may also be separated by freezing out the neon with liquid hydrogen. Argon, which has a liquefaction temperature lying between those of nitrogen and oxygen may be liquefied along with those two elements and separated from them by fractional distillation of the liquid.

CRYOGENIC, OR LOW-TEMPERATURE, RESEARCH

Inception of Cryogenic Laboratory

From what has been said, it is obvious that the production of helium from natural helium-bearing gas involves many scientific and engineering problems concerning the properties and manipulation of gases and liquids at low temperatures and high pressures. When helium production was initiated, the data then existing in this field were scanty, and it early became apparent, as the Government helium project developed, that a laboratory devoted to research along these lines was absolutely essential to success. The Bureau of Mines cryogenic research laboratory was therefore established.

Laboratory Opened by Madame Curie

The laboratory was formally opened in Washington in May 1921 by Madame Curie, the eminent French scientist, codiscoverer of radium, who was in the United States on a visit at that time. This laboratory was ultimately removed to the plant at Amarillo, Tex., and installed in a building constructed expressly to meet its requirements.

Experimental research has been carried on continuously in this unit by a specially trained corps of technical men, and data have been evolved which have been of utmost value to the Government's helium production and purification projects, especially in developing designs for the Amarillo plant. Not only were scientific data evolved in the laboratory, but their practical application was tried out in experimental, laboratory-scale plants, designed for improvement of helium production processes, preliminary to the building of full-scale plants. In this way great economies of time and money were effected.

Work of Cryogenic Laboratory

As now conducted at Amarillo, the work of the cryogenic laboratory includes, in addition to strictly cryogenic research, the supervision of all control analyses pertaining to the production plant and its allied facilities, and performance of analyses relating to the Bureau's continuing gas field investigations. This latter activity comprehends examinations of numerous samples of natural gas, from locations throughout the country, for content of helium and other constituents.

An indispensable adjunct of the cryogenic laboratory is an equipment consisting of compressors and liquefiers with their appurtenances, for producing liquid air, essential in carrying on the low-temperature researches and for

the analyses of natural gas for helium and other constituents. In this apparatus liquid air can be produced in amounts up to about 15 quarts an hour.

In analyzing natural gas for helium content, as done in the cryogenic laboratory, the gas is passed through a system, included in which are glass bulbs filled with a certain variety of specially prepared (activated) charcoal. These bulbs are immersed in Dewar flasks filled with liquid air, at the temperature of which (about -312°F.) all of the constituents, save the helium, are adsorbed and the helium, thus isolated, is collected and its volume measured (62, pp. 41 and 42).

Helium Purification Plants Designed and Built by Bureau of Mines for Army and Navy

Data for the design of plants for purification of helium after use in airships were developed as coming within the scope of the activities of the cryogenic laboratory. A plant was built by the Bureau of Mines for the Naval Air Station, at Lakehurst, N. J. Two plants built by the Bureau for the Air Corps of the Army, one a mobile and the other a stationary unit, are located at the Army Air Station, Scott Field, Belleville, Ill., near St. Louis, Mo. The helium for inflation of the great Navy dirigibles has been purified in the Lakehurst plant.

USES FOR HELIUM

While the outstanding practical, large-scale employment for helium is in floating balloons and airships, other uses for this unique element are being found. It is probable that the number of uses will increase in proportion as more is learned of the properties and adatability of the gas and greater quantities of low-cost helium may become available.

Use in Deep-Sea Diving and Caisson Work

In connection with the use of caissons, also with reference to shaft sinking and tunneling, where operations are conducted under pressures increased above atmospheric for the purpose of excluding water from the working place, the Bureau of Mines as far back as 1922 investigated the subject of controlled oxygen content and greater diffusibility of certain gases (66) in artificial respiratory atmospheres. It was discovered that helium might be used to great advantage in the make-up of such atmospheres, not only in facilitating caisson work, but also in the operations of deep-sea divers, who are obliged to carry on their activities under increased pressure.

Effect of Air Breathed Under Increased Pressure

Air is, roughly, one fifth oxygen and four fifths nitrogen. It contains, however, a number of other constituents, minor in amount, such as water vapor, carbon dioxide, hydrocarbons, nitrous and nitric acids, ammonia, ozone, hydrogen peroxide, and rare gases (67). The blood and other tissues of the body

normally contain some air, but if they are subjected to air pressure above atmospheric, as, for instance, when a diver descends into the sea or when a man is at work in a caisson, increased amounts of air are taken up. If the pressure is great, so much air, and particularly the nitrogen portion of the air, is absorbed that on return of caisson workers or divers to a normal environment, unless extreme care is observed and much time allowed for the change of condition (the so-called "decompression" period), the absorbed nitrogen will collect as bubbles, and these, emitted from the blood and tissues, will cause a pathological condition known as the "bends", which in some instances may be severe enough to cause death.

Effect of Artificial Helium-Oxygen Atmospheres

An exhaustive series of experiments by the United States Bureau of Mines, in cooperation with the United States Public Health Service and the Navy Department (68), has demonstrated that through the use of artificial breathing atmospheres in which helium is substituted for nitrogen and the proportion of oxygen is somewhat reduced, divers and caisoon workers can work for more extended periods and under higher pressures with greatly increased safety and comfort, and may be returned to normal pressure conditions in a fraction of the time required when compressed air is the atmosphere breathed while at work. The pressure-illness and hazard to life from this source of danger are thereby practically eliminated. This is because helium at pressures not far removed from the ordinary is not only inert but is extremely insoluble, being, next to neon, the least soluble of gases. Also its molecule is much smaller than that of nitrogen, and therefore its diffusion is more rapid.

Various Uses Suggested in the Arts and in Medicine

Uses for helium may be found in metallurgical processes, by reason of its inertness and of its comparative insolubility in metals, either molten or otherwise. On account of its inertness, hot metals may be worked in an atmosphere of helium without suffering corrosion or gas occlusions. The following suggestions which have been made for other practical applications for helium might be mentioned: As a preservative for food; in connection with refrigeration processes, by reason of its high conducting power for heat; for cooling electric motors, and for fireproofing high-tension switch boxes, on account of its inertness and high heat conductivity. Furthermore, the use of helium may be indicated in certain medical and surgical applications, according to the results of experimental researches already carried out.

Scientific Uses

There are several ways in which helium has been of service in the research laboratory as an aid to the scientist in his never-ending quest for facts and reasons underlying Nature's phenomena. It finds application in thermometry, for recording extremely low temperatures. It has also found use in spectroscopy as a primary light standard, and to demonstrate properties of electric waves (69). Just recently, it has been proposed to refer all atomic weights of the elements to helium, the helium atom being taken as 4, in place of the usual standards, hydrogen as 1, or oxygen as 16 (70).

Liquid Helium and Electrical Superconductivity

With helium as the instrumentality, the remarkable property of electrical supraconductivity or superconductivity was discovered. It was Kamerlingh Onnes who made the discovery while experimenting with liquid helium in 1911. Liquid helium, it will be remembered, boils under atmospheric pressure at 4.20° K. Onnes found that mercury cooled by liquid helium suddenly lost all resistance to the passage of electric current at 4.19°K, becoming a superconductor (71). Certain other metals, alloys and metallic compounds, have also been found to become sperconductors, or perfect conductors, at various respective temperatures peculiar to each (called "transition" temperatures), in the general region of the temperature of boiling helium, or below. For instance, for titanium and gallium this temperature is at the ultra-low point of 1.1° K. As long as these temperatures are maintained electric currents started in the superconductors persist apparently in undiminished intensity.

Singularly enough, some metals classed under ordinary conditions as poor conductors, such as mercury, lead, and tin, possess this astonishing property of superconductivity, while some excellent conductors, such as pure silver, copper, and gold have so far failed to show it. Then there is the case of copper sulphide, which although composed of a nonsuperconductor, combined with an element which is actually one of the best nonconductors, or electrical insulators, becomes, strangely enough, superconducting at 1.6°K.

No satisfactory theory has yet been advanced in explanation of superconductivity. Judging from facts already developed, it has been surmised that, placed under proper conditions, not as yet however fully recognized, all metals will probably be found superconducting. How far-reaching the discovery of superconductivity will prove to be cannot now be foreseen (72).

Uses in General

In conclusion, it may be said that wherever a perfectly inert gas, or a buoyant gas, or one having high conducting power for heat and electricity, or low solubility, is desired, helium may find application.

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I. C. 6746 October, 1933

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

A REVIEW OF COAL-MINE FATALITIES IN INDIANA DURING THE FISCAL YEAR, OCTOBER 1, 1931, TO SEPTEMBER 30, 1932

BY

C. A. Herbert

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A REVIEW OF COAL-MINE FATALITIES IN INDIANA DURING THE FISCAL YEAR, OCTOBER 1, 1931, to SEPTEMBER 30, 1932 1/

By C. A. Herbert 2/

During the fiscal year ended September 30, 1932, there were 27 fatalities in the coal mines of Indiana. Four of this number occurred in small mines employing less than 10 men, over which the State Inspection Department has no jurisdiction. In addition to these coal-mine fatalities, an explosives accident in a clay mine resulted fatally to one man.

Tables 1 and 2, which follow, list the fatalities as to cause and occupation.

Table 1. - Fatal accidents by cause, fiscal year ended September 30, 1932

Cause (underground)	Number	Percent of total
Falls of roof	11	40.8
Falls of coal	--	--
Mine cars	3	11.1
Locomotives	1	3.7
Electricity	5	18.5
Gas and dust explosions	1	3.7
Mine fires	--	--
Explosives	--	--
Mining machines	1	3.7
Loading machines	--	--
Suffocation (other than fires and explosions)	--	--
Animals	--	--
Falling objects	--	--
Miscellaneous	2	7.4
Shaft	--	--
Falling down shaft	1	3.7
Cage	2	7.4
Material falling down shaft	--	--
Miscellaneous	--	--
Surface	--	--
Total	27	100.0

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2/ District engineer, U. S. Bureau of Mines Safety Station, Vincennes, Ind.

Table 2. - Fatal accidents by occupation, fiscal year ended September 30, 1932

Occupation	Number	Percent	Approximate percentage of employees
Miner	9	33.4+	50.1
Mining machinemen	3	11.1+	5.1
Loading machinemen	1	3.7	4.3
Motorman	1	3.7	3.8
Triprider	1	3.7	3.4
Driver	3	11.1+	2.0
Tracklayer	-	--	--
Laborer	-	--	--
Trapper	-	--	--
Superintendent or owner	2	7.4	.5
Mine boss	1	3.7	.5
Room boss	1	3.7	.8
Motor or parting boss	-	--	--
Fire boss	-	--	--
Surveyor	-	--	--
Shotfirer	-	--	--
Timberman	3	11.1	1.5
Shotrunner	-	--	--
Cager	-	--	--
Electrician	1	3.7	1.5
Pumper	-	--	--
Driller	-	--	--
Miscellaneous	-	--	--
Top labor	1	3.7	8.1
Total	27	100.0	

Referring to table 1 it will be observed that falls of roof ranks first, with 40.8 percent of the fatalities; electricity ranks second, with 18.5 percent; and mine cars third, with 11.1 percent of the fatalities.

By referring to the last columns of table 2 it will be seen that the fatality rate among superintendents or owners of mines for the fiscal year was considerably higher than for any other classification, considering the number engaged; this was because the owners of two small mines received fatal accidents. Mine bosses and timbermen ranked second, drivers third, and room bosses fourth.

Through the courtesy of A. G. Wilson, chief mine inspector, the fatal accident reports by the deputy mine inspectors for the fiscal year ended September 30, 1932, were reviewed, as well as the evidence brought out at the coroner's inquests, and the information thus obtained has been summarized for each fatality, as follows:

Fatality 1. - The front wheels of a car in which a driver on the night shift was riding, jumped the track. The car struck the wooden leg supporting

one end of a railroad iron crossbar, knocked the leg out, and caused the bar to fall. When the man was found, his head was hanging over the end of the car and the railroad rail was resting upon his neck; the steel bar was also resting on a bare machine wire hung along the side of the entry and carrying 250 volts d.c. The workman, who removed the bar from the man's neck, stated that it was charged with electricity, and the report gives the fatality as due to electric shock, although in the absence of definite information to the contrary it is possible that the man may have been choked to death by the bar pressing his neck down on the end of the car. The car itself was of wood and bar-iron construction and was dry.

The report contains no evidence as to what may have caused the car to jump the track--whether it was due to defective track or car, or to some material on the rail, or other cause. There was little clearance on the one side of the track, however, and this points to the desirability of having sufficient clearance where possible, and also of securing the legs, either by cross bracing or by recessing them into the coal rib, so that they will be easily knocked out.

To prevent a recurrence of similar accidents in the future, this company has abandoned the use of wooden legs under steel crossbars and is now drilling hitches into the ribs in which the steel bars are set.

<u>Fatality 2</u>. - The hand of a miner, who was assisting the machine runner in pulling a machine cable into the working place, accidentally came in contact with a bare place on the cable. The man was standing on the rail at the time and received a shock which made him unconscious. The machine cable nips were pulled off the power line as promptly as possible, and artificial respiration was administered, but he could not be revived. His left hand was burned where it came in contact with the bare place on the machine cable.

More careful inspection and repair of trailing cables and the wearing of safety shoes with soles of nonconducting material would probably have saved this man's life.

<u>Fatality 3</u>. - A miner, while attempting to split the end of a piece of fuse with a drill bit, cut the end of his thumb slightly. He paid no attention to the injury other than to wrap the thumb with a piece of friction tape. The next day while helping to move a mining machine, the bar that he was using slipped and he fell up against the coal rib, bruising his shoulder. On the fourth day following the cutting of his thumb, he was obliged to quit work because of pain in his arm and shoulder which he supposed was due to rheumatism. Blood poisoning developed and he died two weeks after receiving the first injury.

This type of accident points to the desirability of reporting all accidents at the time they occur, irrespective of whether they may be considered trivial, and applying first-aid treatment. If the shots in this mine had been fired electrically, this particular accident probably would not have occurred.

Fatalities 4 and 5. - Two men while loading coal at the face of an entry were killed by a large fall of rock. The rock was approximately 9 feet long by 11 feet wide by 12 inches thick and weighed about 4 or 5 tons. There was a water slip at the face, and the loading out of the broken coal had removed the support from one side of the rock, permitting it to fall; the rock was of weak texture and broke into a number of pieces when it fell.

The roof in this section of the mine is generally good, and the nearest timber to the face of the entry where the fatalities occurred was back 26 feet. The mine foreman and part owner of the mine had visited this working place shortly before the men started to work in it and had loaded out some loose rock that had been taken down on the last day the mine worked, which was 4 days before the accident. At the time he loaded out this rock he sounded the roof with a pick and thought it to be safe.

Without question this mass of rock was loose at the time the boss sounded it, but due to the fact that it was of a soft clod nature and of such large quantity, sounding alone with a pick might give little indication as to its actual condition. However, had he used the vibration method of testing he doubtless would have felt vibration in the piece of rock while striking it with the pick and probably would then have taken the necessary precautions to make the place safe. The vibration method of testing is simply holding the tips of the fingers of one hand against the rock to be tested while it is struck with a pick or tool in the other hand.

Unquestionably the two men who were killed failed to inspect the place carefully before they started to work or during the period while at work, otherwise they would have seen the danger and would either have taken the rock down or made it safe by timbering.

Fatality 6. - While pushing an empty car on a parting, a driver was instantly killed by a fall of rock.

The piece of rock that fell was about 4 feet long by 3 feet wide by 1 foot 4 inches thick at the thickest place. The parting on which the accident occurred is 109 feet long and 12½ feet wide and no timbering had been done on it with the exception of a single bar near one end. The top throughout the parting appeared to be good. The report does not state when the roof was last inspected, but presumably it had not been carefully inspected during the previous shift. Track partings are usually given little inspection or attention, and it is surprising that accidents of this type do not occur frequently.

Fatality 7. - A machineman riding in an empty car behind a locomotive jumped out of the car when the trip ran away and in so doing, stumbled and fell under the wheels of the car, receiving injuries from which he died.

The locomotive had been left at the shaft bottom for repairs to the controller and brakes. The motor repairman had made some minor repairs to the controller, had gone to the surface for repair parts for the brakes, and had intended to test the locomotive before it was placed in service. Meanwhile the motor boss came to the shaft bottom to get a locomotive to help push the

main-line empty trip up a hill. Finding the locomotive that was being repaired in front of that he ordinarily used for this purpose he took it instead, not knowing its condition, as there were no markings to indicate that it was in bad order.

While pushing the empty trip, and just before reaching the parting, a wheel came off the fourth car from the end of the trip. The trip was stopped, the last four cars were cut off, and the main-line locomotive continued on into the parting with the balance of the cars. The motor boss, with the aid of a trapper, was trying to put the wheel back on the car when two machinemen, who had finished their cutting earlier than usual and were on their way to the shaft bottom, came along and helped him with it. He then offered them a ride, telling them to get into the empty car next to the locomotive, and started out for the shaft bottom, following the main-line loaded trip that had just passed. They had gone only a short distance and were traveling at rather high speed when the controller arched and flamed, presumably as the motor boss moved it over onto the last notch. He made no attempt to shut off the motor or to pull off the trolley wheel for fear of being burned from the electric arc; instead he jumped from the moving locomotive, and received some minor injuries to one foot.

The machinemen riding in the empty car behind the locomotive, seeing the flash and the motor boss jump, became frightened and jumped from the car. The first man jumped into a shelter hole and was not injured. The second man, however, jumped off between two shelter holes where there was only the average amount of clearance, and stumbled and fell back under the cars, receiving injuries from which he died. There were 5 feet of clearance between the rail and the rib at the point where the second man jumped, but there was a piece of rail and some gob material on the road at this point, and it is probable that he stumbled over the rail or the refuse material.

This fatality was due to several errors or failure on the part of either the company or employees. First of all the motor repairman should have marked the locomotive "bad order" as soon as he took it to the bottom for repairs, and in this case a suitable tag should have been tied to the controller to make sure that no one would attempt to use it while he was on top for repair parts. The company should have some definite system in marking bad-order equipment and releasing it upon completion of repairs.

Second, the motor boss was apparently traveling at a greater speed than was safe while transporting men.

Third, apparently the motor boss lost his head when the controller flamed and jumped off the locomotive instead of pulling the trolley off the wire and applying the brakes.

Fourth, the refuse material and the rail along the side of the entry probably caused the man to stumble and fall under the car wheels, which points clearly to the danger of stumbling hazards such as timbers, rails, refuse, or similar material along the sides of haulage roads.

Fatality 8. - A mine foreman was killed upon being struck by a moving empty car near the shaft bottom and was thrown into a moving trip of empty cars on an adjoining track.

The bottom arrangement at this mine is the usual one for this field where self-dumping cages are used. The loaded cars are caged on one side and the empty cars are run by gravity from the cage to the storage tracks. There is a double crossover switch or diamond on the bottom on either side of the shaft so that both empty and loaded cars may be switched to either track.

In this instance, since a trip of empty cars was being pulled out of the left storage track, the cars coming from both cages were probably being thrown onto the right track where the boss was standing. Just why he stood on the track, knowing that the cars were being dropped down into storage, is not known. Doubtless he lost sight of the fact that the crossover switch had been thrown so that cars from both cages would pass down onto the track on which he was standing. There was ample clearance between the tracks at this point, and if he had stood between the tracks instead of on them he would not have been hit.

There should be a rule at this mine, as well as at all other mines, against standing or walking on the track or either side of the shaft bottom, and the rule should be rigidly enforced; the foreman should set the example by living up to the rule himself, and caution signs should be posted calling attention to this rule.

Fatality 9. - A triprider was caught between a car and a locomotive while making a flying switch onto a parting.

It was the custom for the motorman at this mine to cut the locomotive loose from the empty trip when approaching the parting and to run ahead with the locomotive onto the loaded track, depending upon the switchman at the parting to throw the switch to the empty track for the oncoming empty cars. On the day of the accident the main-line trip was blocked by a gathering locomotive on the parting and the motorman was obliged to stop his trip and wait for the gathering locomotive to "get in the clear". While waiting, his triprider walked ahead to the loaded trip and waited there for the main-line locomotive.

After receiving the signal to proceed the motorman cut off the empty cars at the usual point and ran the locomotive ahead to the loaded trip, expecting the switchman to throw the switch to the empty track after the locomotive had passed, as was the custom. There were 4 or 5 men at the switch as he passed, and he evidently supposed one was the switchman and would throw the switch. In this instance, however, no one had thrown the switch, and just as he was slowing the locomotive toward contact with the loaded trip so that the triprider could make the coupling the empty trip that had followed in on the loaded track struck the locomotive and crushed the triprider between the

locomotive and the loaded cars. The report does not state whether or not the switchman was at his usual post or why the switch was not thrown in accordance with the usual custom.

Flying switches have been responsible for many accidents. They could be avoided if three-track partings were used in the place of those with two tracks; this would give a free passing track and permit the motorman to back his cars, either loaded or empty, onto the parting. Where this is not possible and it is necessary to make flying switches it would be safer to use a switch electrically controlled and operated by the motorman, with a signal light on the switch stand to show which way the switch is thrown; in this way responsibility for throwing the switch would be entirely upon the motorman, and he would have definite knowledge of the position to which the switch had been thrown. Such switches have been on the market for several years and have proved satisfactory. Flying switches should not be allowed in or around any mine because they are decidedly unsafe and entirely unnecessary.

<u>Fatality 10</u>. - A timberman was killed by a fall of rock while standing on the entry approximately 60 feet back from the face.

The roof in this mine requires close timbering. At the time of the accident the timberman had been setting a crossbar near the face and had gone back along the entry to get some cap pieces with which to wedge the crossbar. While he was standing talking to another workman a piece of rock approximately 4 by 5 feet fell out between two crossbars, striking the timberman and killing him.

The report does not state when the roof along the entry had been last inspected. It probably had not been closely inspected during the previous shift or on the shift on which the accident occurred. The area at or near the face is usually supposed to be kept under careful supervision by the workman, but the vast expanse of open territory away from the face frequently receives little or no attention. It is fortunate that so few accidents occur in these places away from the faces.

<u>Fatality 11</u>. - A miner 75 years of age received injuries from a fall of rock from which he died a week later.

The report states that this man had drilled a shot hole up into the roof, which ordinarily was good, and had thus loosened a piece of slate of about 3 inches in maximum thickness and approximately 5 feet wide and 8 feet long. The rock was apparently supported by some top coal. The man who was killed and his partner had just finished loading a car of coal. The partner went back to the crosscut to eat his lunch while the other man remained at the face to pull down the loose piece of slate. The partner is said to have cautioned the man to set some posts under the loose rock to protect himself while taking down the top coal; however, he did not set the posts but stood under the rock while working down the supporting top coal. As soon as the support afforded by the top coal was removed, the rock fell and crushed the

miner. The report further states that the side of the room on which this miner was working was not timbered as close to the face as was the other side. The room boss, who was in an adjoining room, heard the rock fall and the man call for help. The report does not state whether or not the boss had visited this place prior to the accident on the day that it occurred.

The victim, being 75 years of age, no doubt had more than the average years of experience. The fact that he had drilled into the roof and thus weakened it and that his timbering lagged behind his partner's on the other side of the room would indicate carelessness on his part and also lack of supervision on the part of the company. He undoubtedly was aware of the danger in proceeding to do the work as he did, and particularly so since he had been cautioned by his partner to set safety posts under the loose rock before starting the work. It is possible that because of his advanced age he may have had difficulty in keeping up with his partner, who was a younger man, and for this reason he had taken chances while doing his work that he would not have taken otherwise. Men of advanced years should be placed in a section of the mine where the turn is not as good as it is in other sections, and it would appear that all men in a place should work together at least insofar as keeping the entire place in a relatively safe condition is concerned.

Fatality 12. - A miner became sick while wedging down some coal that had been shot the night before. His partners, who were working with him, told him to go back from the face a little distance and rest while they finished loading the car. After the car was loaded these two men went back to see him, and he told them that he was still too sick to work; he went to the surface on their advice while they remained in the working place during the balance of the day and apparently felt no illness as a result of working in the place alleged to have bad air.

The next day the man who was ill died of pneumonia; the autopsy gave the cause of his death as acute lobar pneumonia resulting from breathing carbon monoxide. Inasmuch as the two men who were working with this man felt no ill effects from carbon monoxide or any other cause and since the place was shot the evening before and at least 12 hours had elapsed from the time the shots were fired until the man started to work in the morning, it is questionable whether the carbon monoxide remaining in the place as the result of shooting the night before was sufficient to have brought on pneumonia.

The coal at this mine is all undercut by machine and is shot with black powder and fuse by shotfirers at the end of the shift. Air samples were collected in this working place and showed normal mine air and no carbon monoxide. However, samples of air collected on the day following that on which the alleged gassing occurred would give little evidence as to the condition on the day previous, as the mine is fairly well-ventilated and any carbon monoxide or other gas, if present, would be very likely to have disappeared by the time the samples were taken.

The report does not state how the coroner's physician arrived at his conclusion as to carbon monoxide poisoning. The coroner's report merely states that he arrived at his conclusion as a result of the autopsy and other evidence obtained.

Fatality 13. - At a small mine employing fewer than 10 men and therefore not under the State mining law the cage at the hoisting shaft did not come up into the landing properly when a car of coal was hoisted. It was impossible to pull the car off the cage and the owner and an employee unloaded the car while it was still on the cage. After unloading the car the owner got into it while it was still on the cage and told the hoisting engineer to pull the cage up so that he could see where it was binding or what was preventing it from coming up into the landing. As the hoisting engineer caused the engine to pull on the rope the latter slipped through the clips by which it was attached to the cage, and the cage fell to the bottom of the shaft, killing the owner of the mine.

It is said that the rope had been attached by five clips; this is an ample number and had they been fastened tightly, without doubt they would have held the rope securely to the cage. However, probably the bolts in the clips had worked loose, and when the extra stress was put on the rope in an endeavor to pull the cage up to the landing the rope had slipped through the clips, permitting the cage to fall. Doubtless previous jerkings on the rope had already caused it to slip so that only a small additional pull was necessary to cause the rope to come loose.

Without question this accident would not have happened had the cage and rope been properly inspected as required by law at mines employing more than 10 men. Apparently the cage was not equipped with safety catches to hold the cage if the rope broke or came loose from the cage; the mining law provides that, where more than 10 men are employed, all cages on which men ride shall be so safeguarded. Frequently small mines are inadequately equipped and operated insofar as safety is concerned.

Fatalities 14 and 15. - A room boss and a timberman were instantly killed by a fall of slate at the face of an entry while attempting to set a crossbar.

The fallen rock was 30 feet long, 9 feet 6 inches wide, and 5 feet 7 inches in maximum thickness. The rock had been supported outby the point where they were attempting to set the bar by three crossbars 8 to 10 feet apart. The men evidently knew of the danger but had not taken the necessary precaution to set safety posts to furnish support to the rock while the crossbar was being set. There is also no question but that too much dependence was placed on the three crossbars that supported this large mass of loose rock. The rock had apparently swung the other bars out of place, letting the entire mass fall.

This accident was unquestionably due either to carelessness or to lack of knowledge on the part of the face boss as to how work of this kind might

be done with reasonable safety; it points to the desirability of having definite standards for the setting of safety posts to offer the necessary protection while permanent timbering is being done under dangerous rock.

Fatality 16. - An assistant electrician was electrocuted while hanging an electric cable carrying 240 volts a.c. The cable had been knocked down during the day shift.

Before beginning the work the chief electrician had instructed the assistant to pull out the switch and cut the current off the cable, and he had done so at the time. The switch was only 8 feet from the place where the electrician was working, and for some unexplained reason he threw it in again after he had started to work. Since the lights at the shaft bottom were connected to the circuit on which he was working it is possible he threw the switch back in to obtain more light than his carbide lamp afforded. The wire stretcher he was using to take the slack out of the cable cut through the insulation, so that as he was standing on the rail and wore hobnail shoes and also as the wire stretcher was made of metal, he formed a direct contact from the electric cable to the rail.

This is a case of direct violation of instructions. The man had been working at this type of work for a number of years and was personally aware of the hazard. Had he been wearing safety shoes with insulated soles rather than the hobnail leather shoes, he might have had sufficient insulation between his body and the rail not to have received the shock.

Fatality 17. - A driver getting on the front end of a moving loaded car was caught between the car and a boulder projecting from the roof and received injuries from which he died about 6 weeks later.

The practice of riding on the front end of loaded cars, although more or less general, is extremely dangerous and is prohibited in many mines, the drivers being required to ride on the back bumper. Abrupt changes of contour in the roof also offer a great hazard on haulage roads. It would pay many mines to go to considerable trouble and expense to remove them.

In this case the boulder either should have been taken down, illuminated by an electric light or whitewashed, or otherwise treated to call attention to the hazard.

Fatality 18. - A timberman was crushed by a fall of rock at the face of a room while preparing to make the place safe by taking down a loose rock; he had apparently neglected to protect himself by setting safety posts before starting to wedge down the rock. The piece of rock that fell was 7 feet long by 5 feet wide and was 8 inches thick at the thickest point.

Definite standards covering the safe way of doing work of this kind should be adopted at all mines and should be rigidly enforced.

Fatality 19. - A miner was instantly killed, and his brother who was working with him loading a car of coal at the face of an entry was seriously injured by a fall of rock.

The accident occurred at 9:00 a.m., just as they were finishing loading the car. The rock was 8 feet wide by 10 feet long and 4 to 5 inches thick and weighed several tons. Doubtless loading out the coal at the face had removed some of the support under the rock, permitting it to fall. The report does not state whether the men had examined the roof themselves before starting to work; neither does it state whether or not the fire boss had examined the place and reported the dangerous condition of the rock following his examination before the beginning of the day shift. Unquestionably the men could have recognized the dangerous condition of the rock had they properly tested it before starting to work. If they had done this and knew it was loose they may have been working under the mistaken idea that they could tell how long it would be safe to work under it before it fell.

There should be a definite rule requiring men to examine their roof carefully before beginning work and also requiring the setting of safety posts at the face before beginning to load the coal. Had such a rule been in force and obeyed this accident might have been prevented.

Fatality 20. - A machineman was killed by a fall of rock which occurred just after he had finished cutting the face of the room and was preparing to cut a crosscut a short distance back from the face.

It had been known that the rock was dangerous, and a timberman was in the place where the accident occurred at the time and had just taken a measurement before cutting a post to make it secure.

This accident was due to the fairly prevalent fallacy of permitting men to work under dangerous roof, with the idea that its fall may not take place immediately, and could have been avoided if rules had been enforced and the men required to stay out from under the loose rock until it had been made secure.

Fatality 21. - A miner at a small mine employing fewer than 10 men was killed by electric shock when he slipped and fell on a bare machine wire hung along the side entry.

The entry at this point was 24 feet wide, but because of gob material along the side of the entry the wires had been hung on posts set only 19 inches from the rail. Had the wires been set back at a safe distance this accident might not have occurred. The report further states that the workman who extricated the man from the wires merely fanned him until additional help came, although at the time he was removed from the wires he was still gasping for breath. No attempt was made by anyone to administer artificial respiration, and when the doctor arrived the man was pronounced dead. It is possible in this instance that had the man who took him off the wire known how to administer artificial respiration the victim might have been saved.

With the common use of electricity in mines, even small ones employing fewer than 10 men, it is very important that all workmen have some knowledge of first aid and particularly the application of artificial respiration. The use of bare power lines in mines without protecting them, while more or less universal practice, is dangerous and should not be allowed.

Fatality 22. - Two topmen pulled a car of dirt from a cage at the dirt landing in the tipple and after unloading it pushed it back to place it on the cage, but they did not observe whether the cage was still at the landing.

In this instance the cage had been lowered to the bottom of the shaft, and instead of pushing the car onto the cage the men pushed it into the shaft, one of the men falling with it and being instantly killed. The report states that there was no gate at this landing but that after the accident a gate had been installed. All shaft landings should be amply protected by gates and fencing to prevent men from falling into the shaft or from pushing empty cars or allowing material to fall into the shaft. Each year there are many accidents of this kind in which the men, thinking that the cage was still at the landing, have pushed cars into the shaft where no gates had been installed or where the gates had been left open; and when gates are placed they are of little use unless they are kept closed at all times, except when cars are being put on or off the cage or when men are getting on or off the cage. At many mines gates are installed that are automatically kept closed, except when the cage is at the landing.

Fatality 23. - A pumpman and general utility man was killed by having his leg caught in the propeller chain of a moving machine truck.

The man was moving a mining machine from a room out onto the entry, and just as he passed over the knuckle of a rather steep down grade the power went off the machine, which began to coast downhill toward a mule a driver had left standing on the entry a short distance ahead. Apparently the man, thinking he would not be able to stop the machine in time, had stepped off the truck to avoid hitting the mule; but owing to the fact that the track at this point had been lowered into the fire clay bottom, leaving a sloping shoulder on each side of the track which was damp and slippery, the man in stepping off onto this shoulder lost his footing and one leg was caught in the propeller chain. About the time the man stepped off the truck, the pit boss who was standing close by saw there was danger that the truck would hit the mule and got it out of the way. In his testimony the mule driver, who had left the mule standing on the entry, stated that he had seen the machine cable on the track and to avoid running over it with a car on his way out had gone back and taken the nips off the power wire, apparently just about the time the man started down the hill with the mining machine.

Unquestionably this accident could have been avoided had the entry been leveled off when the track was cut down into the bottom and thus eliminated the slipping hazard that caused the man to get his leg into the propeller chain. In addition to this, it is unwise for a driver or anyone else to take the nips off the trailing cable of a power unit without first notifying the man or men who are operating the unit. In many mines it is now required

that where a cable crosses a haulage road it must be hung overhead to avoid the possibility of being caught by a car or locomotive. Hooks for this purpose are placed in the roof at the desired points.

Fatality 24. - A loading-machine runner was killed by a fall of rock at the face of a room. The man was operating a Joy loading machine and had knocked out some timbers in the way of the machine; shortly after he had taken out the timbers a piece of rock 18 feet long, 18 inches wide, and 9 inches thick fell on him, killing him instantly. His assistant stated that before taking out the timbers both had sounded the top and it seemed to be all right. The timberman stated he had set the timber the day before the accident and that on the morning of the accident, before the loading machine had been taken into the room, he had visited the place and saw that some of the timber had been taken out to make room for the machine. He stated that he had also sounded the roof at that time and it seemed fairly good. Evidently the roof had needed the timbers under it or the timberman would not have set them the day before; and knocking them out without first taking the precaution to set others to take their place, possibly using a crossbar or two, was hazardous.

While both the timberman and assistant loading-machine operator testified that they had sounded the roof, sounding alone might not indicate the looseness of a piece of rock the size of this one--18 feet long, 18 inches wide, and 9 inches thick. The vibration method of testing should be used; that is, resting the tips of the fingers of one hand against the roof while striking the roof with a bar or pick held in the other. The vibrations set up in the loose rock are transmitted to the fingers, thus giving a warning of looseness even though the sound may not indicate danger.

There should be definite rules at all mechanically operated mines requiring that before knocking out timbers, particularly under loose roof, some other means of roof or side support should be afforded; these rules should be rigidly enforced by the company. Where it is impossible to timber in such a way as to get to the coal with the loading machines it would be far better to move the coal by hand to a point where the machine can get it rather than to take the chance of killing someone.

One mine in Indiana is using long steel crossbars where similar conditions prevail.

Fatality 25. - At a small mine employing fewer than 10 men a miner collapsed on the cage while being hoisted as a result of having breathed powder smoke while attempting to load a car of coal.

The man's head was caught between the cage platform and the shaft timbering, breaking his neck. A new shaker screen had just been installed at this mine and the officials were desirous of giving it a trial. Not having any coal prepared that could be loaded into the cars for this trial, the victim and another miner went into the mine and drilled and shot one hole;

after waiting an hour or more both went inside and began to load a car. However, both began to feel sick after working about 15 minutes and went out to the shaft bottom and started up on the cage. When within about 20 feet of the top of the shaft, which is about 60 feet in depth, one of the men collapsed and fell to the platform of the cage with his head toward the shaft curbing. His head was caught between the platform of the cage and the shaft timbering.

This accident shows the hazard of going back too soon after a shot is fired, particularly where black powder is used, as was the case in this instance; and scores of cage accidents have occurred where men have been caught and killed between the cage platform and shaft timbering; they could be prevented if cages were provided with doors or gates which were always closed while men were being hoisted or lowered.

Fatality 26. - The owner of a small mine employing fewer than 10 men was burned to death by the ignition of methane at the bottom of the shaft.

The owner and his son had just been lowered on the cage to examine the mine for gas before starting to work for the day. The son had a Baby Wolf flame safety lamp and states that he found gas as they stepped off the cage. He claims that he then smothered the lamp by placing it under his jacket, after which he placed it on the floor, sat down and was talking with his father. Suddenly there was a flash; the son jumped into the sump and escaped injury, while the father ran down the entry and received burns from which he died.

No definite explanation is offered as to the cause of the ignition. Both men carried carbide lamps filled with carbide, but neither lamp had any water, although one lamp had some dissolved carbide which might indicate that there was some gas being given off by the lamp. A more probable explanation is that the son may have thought he smothered the flame safety lamp by putting it under his jacket when as a matter of fact gas was still burning up in the gauze of the Baby Wolf flame safety lamp. When the burning gas reached a temperature sufficiently high, it may have ignited the explosive mixture.

This mine is working in the Number Five bed and the shaft is about 190 feet in depth. The practice has been to run the fan only on days that the mine worked. Doubtless the fan had not been operating for a considerable period prior to the time the men went into the mine, and they did not wait a sufficient length of time for the mine to be cleared of explosive gas. Hence, they found the explosive gas at the bottom of the hoisting shaft, which was also the up-cast shaft. This incident has numerous lessons; among them it illustrates one of numerous hazards in having the hoisting shaft in return air.

Fatality 27. - A triprider who was acting as a motorman on an idle day was electrocuted by coming in contact with a trolley wire while trying to put the trolley wheel back on the wire.

The man who was working with the victim was at a switch a short distance from the locomotive and saw that the other man was having some trouble getting the trolley back on the wire. Suddenly the motorman called. The man at the switch walked back to the locomotive to see what was the trouble, and found him sitting on the seat in the cab with his arm across the controller with his head resting on his arm. Receiving no answer when he spoke to him, the partner called for help; the room boss at the time was about five or six hundred feet down the entry and in response to the calls ran to assist in taking the motorman off the locomotive. At the time the room boss arrived, the victim was gasping for breath, but apparently neither the boss nor the other workman knew how to give artificial respiration, for they merely rubbed and worked his arms and bathed his face for about 5 minutes after taking him off the locomotive. Not being able to revive him, they loaded him into a car and took him to the shaft bottom and again worked on him until the arrival of a doctor who applied artificial respiration. Later a pulmotor was used on him for an hour and 15 minutes without helpful effect.

In this instance probably as much as half an hour elapsed between the time they found the man gasping for breath after receiving the electric shock and the time the doctor arrived. It is essential that artificial respiration be started as soon as possible after an electric shock, and preferably in less than 10 minutes. There was therefore in this case very little chance for the doctor to be of any assistance, but there is a very strong probability that if these men who brought the victim to the surface had known how to apply artificial respiration and had given it immediately after taking him off the locomotive, his life would have been saved.

With the continual increase in the use of electrical equipment underground it is very essential that those working in the mine should have some knowledge of first aid, and particularly of artificial respiration. In fact every person who works in or around any mine should be familiar with the full first-aid course as given by the United States Bureau of Mines.

Fatality 28. - In addition to the coal-mine fatalities enumerated, a fatality occurred in a clay mine in southern Indiana.

In this mine eight dynamite shots were lighted, but all eight failed to fire. On the succeeding day, finding that the eight shots had misfired, eight additional holes were drilled and shot parallel to those that had failed. On the third day while digging with his pick in the broken material that had been shot down by the second round of holes, a man apparently struck the unexploded primer from one of the misfired shots, causing an explosion which resulted in his death.

A fellow workman, who was about 200 feet distant from the man killed, said that the explosion was quite violent and blew out his light. Possibly more explosive than just the primer exploded when the man's pick struck the detonator.

The report does not give information as to why all eight shots failed to fire, but since fuse was being used it is presumed that failure was due to using deteriorated fuse on all eight shots. Fuse that is allowed to become damp is a fruitful source of misfired shots.

The handling of misfired shots presents many hazards. No criticism is offered as to the method pursued in this case--namely, the drilling and firing of new holes--as it is believed this would be safer than attempting to withdraw the misfired charge. However, after the new holes are fired extreme care should be used in loading out the material shot down to avoid such an occurrence as took place here. At the same time great care should be exercised so that any unexploded primers, "caps", or explosives may be found and not loaded out with the broken material. A very grave hazard exists if unexploded explosives are permitted to pass through the surface plant and possibly ultimately reach the bin or the furnace of a consumer of coal.

CONCLUSION

If careful investigation of an accident is made and the report contains the essential information concerning the underlying causes which led up to it, usually it is not difficult to fix the responsibility for the accident or to determine what might have been done to prevent it. It is not always so easy, however, to recognize and correct these underlying accident causes before the accident occurs, and it is only through a study of the accidents that the necessary foresight or vision may be obtained that will enable mining people to recognize and correct dangerous conditions or practices and thus to take measures to reduce mine accidents.

Some State mine inspection departments now issue monthly mimeographed statements descriptive of the essential factors which entered into the occurrence of the fatal accidents of the coal mines of the State for the month, pointing out what might be done to avoid the accident or accidents. This is a worth-while procedure and one which all State inspection forces might well adopt.

I.C. 6747
October, 1933

INFORMATION CIRCULAR

UNITED STATES BUREAU OF MINES

USE OF ELECTRIC POWER IN CASTLE GATE NO. 2 MINE, UTAH FUEL CO.[1]

By D. J. Parker[2]

INTRODUCTION

Owing to its flexibility, convenience, economy, and availability, electric power is peculiarly adapted to the needs of the coal-mining industry. Because of such obvious advantages, the application of electricity has in recent years become more and more extensive and is now a most important factor in the Nation's coal production.

With the introduction of this type of power into coal mines certain new, specific, and well-recognized hazards resulted. The question arises, therefore: Has the industry met these hazards squarely and effectively? With respect to some individual companies, the answer happily is in the affirmative; with respect to the industry as a whole, there is much room for improvement.

In order successfully to reduce the various hazards, including shock, burns, and gas- and coal-dust ignitions, with attendant fires or explosions, the United States Bureau of Mines has through years of effective research work established certain minimum safety standards relating to the construction of electrical equipment for use in coal mines, which have been met by many progressive manufacturers. As a result there are today available approved types of electrical equipment for almost every known underground use, including transportation, communication, illumination, mining, loading, pumping, etc.

In addition, the Bureau of Mines, in cooperation with mining officials and State mine-inspection departments, has fostered and promoted an intensive educational campaign looking to the elimination of electrical as well as other types of injury to employees of the industry.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6747."

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Electricity causes an average of 4 to 4½ percent of all coal-mine fatalities and ranks fifth in the causes of fatalities in coal mining; these figures do not, however, include fatalities from gas and dust explosions caused by electricity. The number of underground fatalities from the latter cause has increased 25 percent during the past 10 years.

ACKNOWLEDGMENT

The author acknowledges his indebtedness to A. C. Watts, assistant manager, Utah Fuel Co., and to The Calumet Fuel Co. for much of the information on which this paper is based.

GENERAL DESCRIPTION

Castle Gate No. 2 mine is operating in the Castle Gate "D" bed, which averages 24 feet in thickness and dips on a 10 percent grade toward the north. The elevation at the portal is 6,131 feet above sea level. The cover ranges from a few feet to 2,500. This bed occurs in the Book Cliffs field, Mesaverde formation, Upper Cretaceous system.

Approximately 200 men are employed, and the average daily production is 3,000 tons.

Except for some hand loading and limited animal haulage Castle Gate No. 2 mine, under normal operating conditions, is completely electrified.

Because the mine produces more than 300,000 cubic feet of methane every 24 hours, and explosibility of the coal-dust is high, and the coal is easily ignited, unusual precautions have been taken to reduce existing hazards, including the selection, installation, and maintenance of electrical equipment, rock-dusting, adequate ventilation, closed lights, unstinted use of water, and good supervision.

SURFACE EQUIPMENT

Main Transformer Station

Current enters the main transformer station at 44,000 volts and is reduced to 4,000 volts. The station is substantially fenced, and the fence is provided with a gate which is kept locked at all times when not in actual use. Danger signs are posted. Switches, circuit breakers, and other necessary equipment are installed in a nearby concrete building, and rubber mats are kept in front of switch panels.

From this station the 4,000-volt circuit connects with the two outside motor-generator sets. A 4,000-volt cable enters the mine through the first left escapeway off the dips, and branch circuits are taken through the slope manway. A 440-volt cable also enters the mine through a cased drill hole at the top of the raises solely to supply power to the main hoist, located close by. A transformer in a well-guarded surface station near the collar of the drill hole reduces 4,000 volts to 440.

The two motor-generator sets located on the surface produce 500 volts direct current for both surface and underground haulage.

The fan is operated by a 4,000-volt, a. c., 200-hp. motor, belt-connected. It is in a steel housing so arranged as to be well-protected from violence in case of an explosion.

UNDERGROUND TRANSFORMER STATIONS

Power enters the mine at 4,000 volts a. c. through a heavily insulated triple-conductor cable installed on the floor of the escapeway. Branch circuits connect with the various transformer stations. The 4,000-volt current is reduced to 440 and 220 volts. All alternating-current equipment is operated on 220 volts except the slope hoist, which requires 440 volts. The use of direct current is confined entirely to the operation of trolley locomotives.

The 12 transformer stations are at strategic points throughout the mine, and all are on intake air currents. The transformer rooms are of tile, with a concrete roof and floor; the latter is covered with about 6 inches of sand, which serves as an absorbent in case of oil leakage or spillage.

A steel door, so arranged that it will close by gravity and held open in nearly horizontal position by a hemp rope passing over the principal electrical apparatus, is standard equipment on all stations. This door, when closed, hangs at an angle of about 20° off the vertical, so that its own weight will keep it securely closed should the rope be burned through.

The electrical equipment in these stations consists of three 4,000- to 220-volt a. c. transformers, together with the necessary oil switches and circuit breakers. The transformer casings are well grounded and adequately guarded.

Each transformer station in the mine has an overload-relay switch. Short circuits in any section throw out the switch at the station transformer, and the lines to this station must be inspected and repaired before the switch can be closed. The rest of the mine on other transformer stations is not affected.

Not less than 2 cubic feet of rock-dust in metal containers are kept on the outby or fresh-air side of each station for fire-fighting.

From the transformer stations secondary cables, carrying 220 volts a. c. for the operation of portable equipment, lead to all sections of the mine. When it becomes necessary for the cables, either primary or secondary, to cross haulageways, this is accomplished by passing the cables under the tracks and through conduits consisting of short sections of 3-inch pipe.

All cables are laid close to the rib on the floor and are not suspended. They are protected from injury wherever they might be subjected to mechanical wear.

The roof throughout virtually all parts of the mine where cables are laid is coal, and there is little or no sloughing or falls to damage the cables. If the roof were bad the cables would either be laid in ditches or suspended. It is preferable to bury them when there is any danger from roof falls.

CONNECTING TRAILING CABLES TO SOURCE OF POWER

It is almost general practice in mines where direct current is used for cutting and drilling to connect the positive conductor to the trolley wire by means of a hook or nip, the negative conductor being hooked to the rail. In moving a mining machine from one working place to another it is usual in coal mining to slide the nip or hook along the trolley wire to save time, and this practice of "nipping" mining machines has been responsible for many disastrous explosions. There is a record of 5 explosions from this cause in 1 mine, 3 occurring within 3 years, the last 2 less than 5 months apart.

Where alternating current is used it is customary in practically all mines, those of the Utah Fuel Co. excepted, to remove the insulation from a small area on the three separate conductors opposite each room neck to fasten the nips of the trailing cables thereto. Where this system is employed, short pieces of $\frac{3}{4}$-inch garden hose are sometimes placed on the conductors so that they may cover the bare places on them when trailing cables are not attached; this is the practice in some Utah mines. Numerous serious explosions resulting in fatalities have occurred from arcs caused when disconnecting the nips from the power circuit. The use of permissible junction boxes would eliminate, or at any rate greatly reduce, both the shock and the explosion hazard.

In Castle Gate No. 2 mine, where alternating current only is used for operation of portable equipment other than locomotives, the trailing cables of loading and mining machines and drills are connected to the secondary cables by means of "arctite circuit-breaking plug receptacles". These receptacles, the capacity of which is 600 volts-175 amperes, are so constructed that any arcing that may take place while the trailing cables are being connected or disconnected occurs before the sleeves of the male and female connections are separated; that is, the arcing occurs in a tightly enclosed space. Moreover, since the receptacles are close to the floor, if any arcing should occur, it is unlikely that accumulated explosive gas would be found close to the floor in an air course, usually in intake air.

Another desirable feature in connection with the secondary cables is a ground conductor in the form of a wire netting enclosed in the cable and connected to a circuit breaker in the transformer station.

The female section of the receptable is spliced into the primary cable at regular intervals, usually opposite every other room neck. The primary cable is carried along the floor on the opposite side of the entry from which rooms are turned.

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This system of supplying power to portable machines has proved to be safe because it eliminates open circuits, thereby reducing the shock and explosion hazards, and it is economical in that no supporting timber or insulators are required. It is rarely necessary to use timber along haulageways to support the roof material in advance workings.

DIRECT CURRENT

Castle Gate No. 2 mine is one of the few remaining mines of the west in which 500 volts are used. Storage-battery locomotives are used for gathering purposes to a limited extent.

The Utah Fuel Co. was the pioneer company in this State and introduced electrically operated locomotives into its mines about 1891. Since 1891 this company has operated at one time or another 8 mines in Utah and 1 in Colorado, all using 500-volt, d. c. locomotives, yet has had only one fatality from electricity (during the Sunnyside mine fire in 1921); this fatality resulted when a miner attempted to board the man-trip before the power was cut off the trolley wire.

In accordance with available information, the Utah Fuel Co. has undoubtedly established a record that might well be emulated, in that it has operated several rather thoroughly electrified coal mines over a period of 42 years with but one fatality from electricity and with no fatality from this cause from 1891 to 1921, a period of 29 years.

Incidentally, this company for a number of years was essentially a training school for mine officials, as evidenced by the fact that, as additional companies were formed in the district, mine foremen and superintendents in numerous instances were drafted from its organization.

The greater hazards involved, due to the use of 500 volts as against 250 or 275 volts on the trolley circuit, are well-recognized. A study of electrical accidents extending over a period of 2 years in one of the principal coal-producing States indicates that the liability of electrocution is approximately $4\frac{1}{2}$ times greater at mines using 500 volts than at mines using 275 volts or less. This study indicates that electrical accidents tend to increase as the square of the voltage; that is to say, if the voltage is doubled, electrical fatalities are likely to be approximately quadrupled for the same number of exposures to the current.

No. 2 mine has never had a fatality from contact with the 500-volt circuit or any other circuit. The principal reasons for this excellent record are: (1) The trolley wire is well-supported and maintained uniformly $6\frac{1}{2}$ feet above the rail. (2) Sectional circuit breakers are installed and used. (3) The power is shut off the trolley wire while the men are loading and unloading from the man trips. (4) All men are instructed regarding shock and fire hazards. (5) Constant and efficient supervision is in effect. (6) By virtue of the hazard itself a wholesome respect is commanded at all times and under all circumstances.

It would appear that the immunity, up to the present, from electrical fatalities in No. 2 mine (when the relatively high voltage on the trolley circuit is taken into consideration) is not due either just to good fortune or good luck, or to any unbalancing of the equilibrium of the law of averages, but rather to well-directed and vigorous effort by the management to educate the personnel of the entire organization to think safety and work accordingly.

The media through which this has been accomplished are (1) monthly mass safety meetings, (2) regular foreman's meetings, (3) 100 percent first-aid training, not once but twice, (4) personal contact with the workmen by the foreman and subforemen, not infrequently, but several times during a single shift, and (5) rigid but fair discipline at all times.

SUGGESTIONS

In the interest of the 80 to 100 or more lives that are at stake annually in the coal mines of the country due to the use of electricity, a few fundamental rules, the application of which is vitally necessary if even reasonable safety is to be attained in the use of electricity in coal mines, will be repeated here.

The following suggestions are therefore offered in the hope that they may serve as an added stimulus to those responsible for the prevention of electrical accidents:

1. Every mine using electricity should be inspected at least once each year by a competent electrical engineer, who should make a written report of observed data on surface and underground installations and practices. The report should cover efficiency as well as safety. Preferably, such engineer should be familiar with underground mining, but not necessarily stationed at or near the mine.

2. In addition, monthly inspections should be made by the local electrician, who should submit a written report.

3. Surface and underground electrical systems should be installed and operated in accordance with suggestions given in U. S. Bureau of Mines Technical Paper 402, Safety Rules For Installing and Using Electrical Equipment in Coal Mines.

4. Because trolley circuits are responsible for most electrical contact accidents, as well as for many fires and explosives in coal mines, it would be desirable to substitute storage-battery locomotives for trolley-type locomotives, especially in gassy mines. In mines where trolley locomotives are used, they should be confined to main haulageways on intake air.

5. Where trolley locomotives are used, the trolley wire should be substantially guarded throughout its entire length wherever the trolley wire is less than 6½ feet above the rail.

6. Trolley wires should be so placed as not to come in contact at any time with inflammable material, such as roof or rib coal, timber, brattice curtains, doors, door frames, etc.

7. Trolley wires should be installed in as straight a line as possible, about 6 inches outside the rail, 6½ feet or more above the rail, where feasible, and on the opposite side of the track from shelter holes or travelways.

8. Signal lines and telephone circuits should be installed on the opposite side of the haulageway from trolley or power circuits.

9. The necessity for maintaining slopes and haulageways, especially those carrying electricity, in a thoroughly rock-dusted manner is emphasized by the fact that several coal-dust explosions have been the result of wrecked trips dislodging the trolley wire or breaking buried transmission cables.

10. Danger signs should be posted conspicuously at points where such warnings will be the most effective in reducing possible contact with live or moving electrical apparatus. Such signs should specify the type of danger.

11. Instructions for the resuscitation of persons overcome by electric shock should be posted in every surface and underground station and hoist room, and also on the principal bulletin board, and all persons employed in or around mines should be kept well trained in methods of giving first aid to the injured.

12. Metal containers of not less than 2 cubic feet capacity should be kept filled with dry sand or rock-dust and located on the outby or fresh-air side of all underground electrical stations for use in case of fire.

13. In addition, fire extinguishers of the carbon-tetrachloride type should be available for combating possible electrical fires. Such extinguishers, however, should not be used in enclosed or confined places where there is little or no ventilation.

14. Incandescent lamps should not be permitted to come in contact with inflammable material. Only weatherproof, keyless-type lamp sockets with no exposed metallic parts, or equipment of equal or greater safety features, should be used.

15. All cables entering mines through boreholes or vertical shafts should be installed in accordance with the best-known practice and in such manner as not to bring unnecessary strains in the covering, insulation, or conductors.

16. All underground power circuits should be provided with sectionalizing switches at intervals not to exceed 2,500 feet. Such switches also should control all branch circuits from the main power lines.

17. It is the practice in some mines to locate power circuits in return airways. Such location of power circuits in gassy mines is decidedly hazardous unless the conductors are carried in conduit or placed below the surface of the floor.

18. All electrical equipment, including conductors, switch gear, etc., should be of the very best material and should be installed in a first-class, businesslike manner.

19. Permissible electrical equipment only should be used, especially in gassy mines.

The foregoing are but a few of the numerous items that may be mentioned as necessary to safe installation and maintenance of electrical equipment. These deal largely with mechanical aspects of electrical hazards; of équal or greater importance is the human factor, which can be dealt with successfully only through education, discipline, and supervision.

CONCLUSIONS

Obviously, the responsibility for the prevention of electrical accidents rests almost entirely on the management rather than on the employee. There is, however, a definite obligation devolving upon the employee to exercise due care in avoiding such accidents as far as is humanly possible. The employer is responsible for the safe and proper installation, maintenance, and guarding of electrical equipment; furthermore, he is responsible for the education of the employee to the extent that carelessness, negligence, indifference, faulty judgment, and ignorance are reduced to the lowest possible degree.

There is probably less excuse for electrical accidents, than for any other cause of injury and death to workmen in the mining industry; the greatest number of fatalities from electric shock are caused by contact with live conductors, nearly all of which can be properly guarded with relative ease.

Our industrial progress and human welfare are based largely on the wise use of electric power. The demand for the application of ordinary common sense to curtail the many hazards incident to the use of electricity in mines has never been so great as it is today.

The progressiveness of the manufacturers has made it possible to procure, at small additional cost, safe, dependable, permissible equipment, the proper use and maintenance of which will go far toward reducing electrical hazards in coal mines.

The use of 4,000-volt highly insulated and protected power lines, as in effect in Castle Gate No. 2 mine, has given no trouble in transmission or leakage of current and has resulted in no accidents to date. The use of the auxiliary highly insulated 220-volt alternating power cables with the enclosed receptacles is both safe and efficient in itself and in addition requires the use of two men rather than one on mining machines, another safety feature very frequently not found in mines.

Unquestionably, the electrical installation in Castle Gate No. 2 mine is one of the most thoroughly safeguarded electrical installations in any coal mine in the United States insofar as probability of explosion ignition is concerned.

I. C. 6749

October, 1933

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

MARKETS FOR RESIDENTIAL STONE

BY

Paul Hatmaker

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INFORMATION CIRCULAR

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MARKETS FOR RESIDENTIAL STONE[1]

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CONTENTS

INTRODUCTION

A primary function of the Bureau of Mines is to encourage elimination of waste and the most profitable use of mineral raw materials. In line with this objective the Bureau has frequently stressed the importance of waste utilization at stone quarries. From time to time studies both technical and economic in character have been made of problems pertaining to the most efficient use of natural stone, particularly that which may be classed as a by-product of the higher grades of dimension stone.

Every quarry producing ornamental or building stone yields large quantities of material that for one reason or another is unsuited for high-grade products, even though a great deal of work has been expended upon it more or less unavoidably. It is obviously good judgment to sell this odd-sized or off-grade stone at whatever price can be obtained rather than to spend more money to place it on the waste dump. In many cases, however, a really profitable side line has been developed.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6749."

2 Mining engineer, building materials section, U. S. Bureau of Mines.

The field of medium-priced home construction appears to be one of the most attractive outlets for stone of this character. A number of dimension-stone producers have cultivated this market to some extent, but the field offers far greater possibilities than have yet been realized.

Residential construction was at an abnormally low rate during 1932 because of general postponement of building projects, but this condition cannot be expected to continue indefinitely. Some authorities believe, in fact, that residential construction will be one of the first industries to recover. Figure 1 shows the importance of this market and the extent to which residential contracts have declined from the peak years before the depression. Steady growth in population, fire losses, obsolesence, and depreciation are constantly operating to create shortages in housing facilities in most communities; and returning purchasing power, and sound banking and financial conditions tend to convert this accumulated demand into an effective stimulant to the construction industry.

The producer of residential building stone finds many satisfactory building materials on the market with which his product may be used. Among these are wood, stucco, concrete, various artificial stone products, and brick, hollow tile, and other ceramic materials. Each has its legitimate place in residential construction, and all of them are used in combination with stone to produce the great variety of artistic effects popular even in smaller communities.

The purpose of this paper is to outline in a general way the possibilities open to the stone producer or contractor in the residential market. The data given are meant to be suggestive of further lines of study rather than exhaustive in themselves. In preparing this report the author wishes to acknowledge the helpful cooperation of the North Carolina Granite Corporation, the Cleveland Quarries Co., the Indiana Limestone Corporation, and the Briar Hill Stone Co.

THE BUILDING STONES

Virtually any stone that is sound structurally, amenable to shaping, pleasing in appearance, and available to the markets can be utilized for residential construction. The more common natural stones now being offered to the consumer include granite, limestone, marble, sandstone, and slate. Granite, limestone, and sandstone are used extensively for wall construction, marble is used widely as interior trim, and slate is much used as roofing material, although an important use is for flagstones. The usefulness of the various natural stones depends mainly upon inherent physical properties such as durability, strength, color, and texture. Being products of nature's workshop, they all vary in composition and physical characteristics depending upon the geological processes by which they have been made.

Granite is among the most ancient of rocks, formed ages ago by slow cooling of molten mineral matter under pressure and now exposed for use through long erosion. It consists chiefly of interlocked quartz, feldspar, and mica

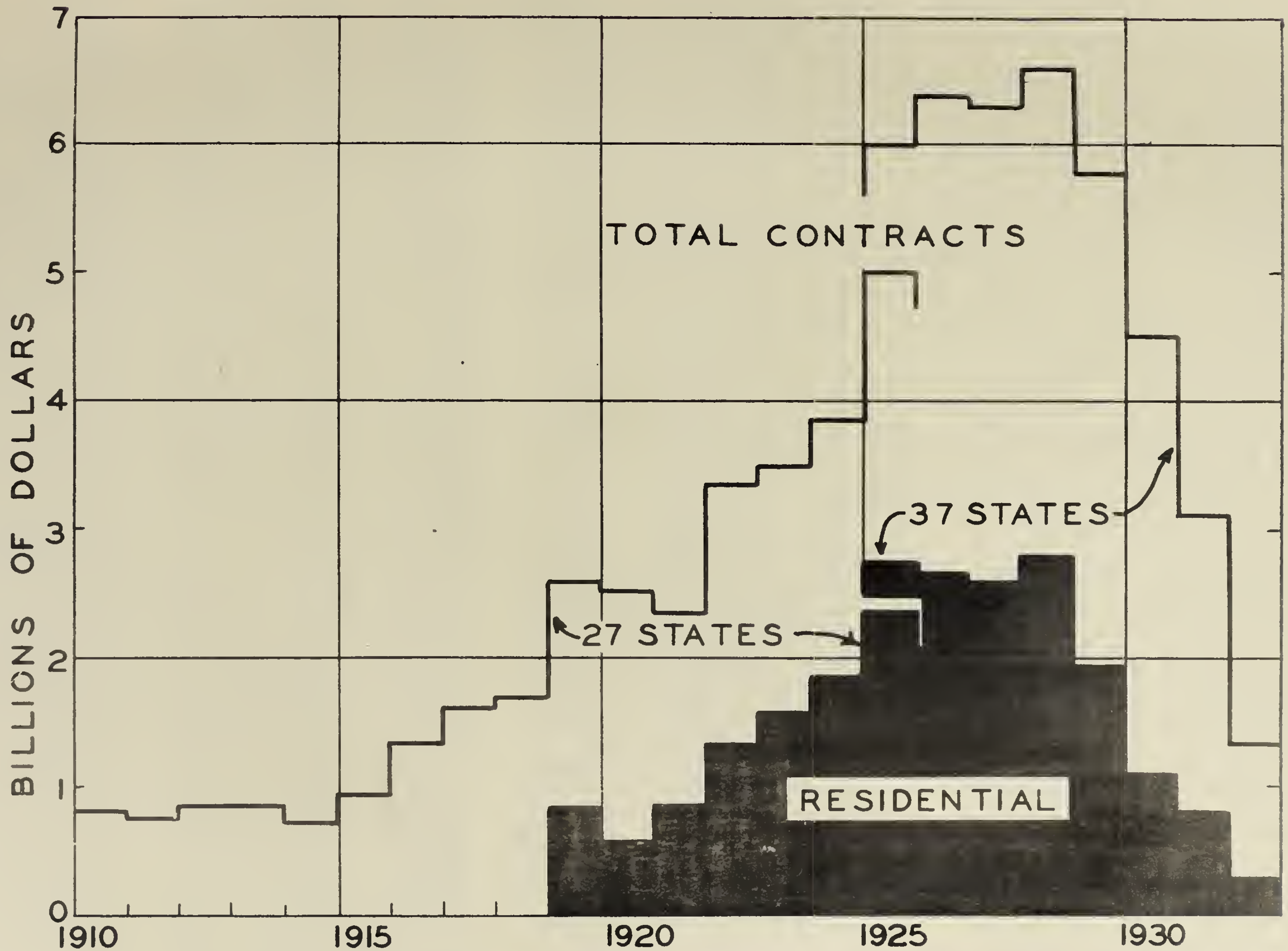

Figure 1.—Construction contracts awarded in 27 States from 1910 to 1925 and in 37 States from 1925 to 1932. (Data from F. W. Dodge Corporation.)

crystals firmly welded together. Shades of gray and pink or red are the most common color effects, the quartz usually being transparent to gray, the feldspar gray to different shades of pink or red, and the mica relieving the ground mass with darker shades of gray to black or sometimes yellow coloring. The texture may be fine- or coarse-grained. Sound granite is strong structurally, is dense and therefore relatively impervious to water, and is very durable. Granite as a commercial term includes not only the true granite of the geologist, but also certain varieties of metamorphic rocks such as some schists and gneisses. Certain of these stones cleave or break readily along parallel planes, whereas most true granites have less tendency to break in well-defined directions, although they may be fractured along the rift and grain. The rift is the direction of easiest splitting; the grain, at right angles to the rift, is less easy to split than the rift yet easier than the "hard way" which presents the greatest resistance to breakage and is at right angles to both rift and grain.

Limestone is a term broadly applied to sedimentary rocks composed essentially of calcium carbonate with or without magnesium carbonate. Most of the popular building limestones consist chiefly of calcium carbonate. Limestones vary in color from pure white to black, but those most used for building vary through shades of cream, buff, or gray. The texture may be fine-grained to coarse-grained, some varieties being distinctly crystalline. Limestone is a relatively soft material yet possesses ample strength and marked durability. Some limestones occur with many bedding planes of varying thicknesses but other varieties are homogeneous and break with equal facility in any direction.

Marble is a recrystallized limestone, which has become dense and relatively hard through subjection to heat and pressure. Marble polishes readily and is available in many shades of color. Occasionally other rocks, notably serpentine, have similar physical properties and are commercially classed as marble.

Travertine, another variety of limestone, is a product of mineral springs. Its banded and porus texture combined with warm shades of color produce unique and interesting effects.

Sandstone, also a sedimentary rock, is composed mainly of quartz grains cemented together. The color generally is gray, cream, or buff to different shades of brown, pink, or red. Sandstone may be friable if loosely cemented or may be very hard and compact with little or no evidence of bedding planes. When so tightly compacted and cemented as to have lost the structure of the original sand grains it is called quartzite. Most building sandstones are moderately cemented, thus having adequate strength as well as easy workability.

Slate is a rock resulting from mud or clay which has been subjected to such great heat and pressure that the original minerals have altered almost completely to new minerals having tabular crystals rearranged along definite and pronounced cleavage planes. Slate is generally blue black to dark gray in color with light gray, green, and red shades occurring in restricted areas. The texture is fine-grained and uniform, and the material is characterized by the facility with which thin, dense, hard sheets may be prepared.

REQUIREMENTS FOR RESIDENTIAL STONE

No single formula covering desirable physical or chemical properties for residential stone can be set forth which will apply equally to all stones. Certain general requirements, however, may be mentioned.

Strength

Stone construction is always associated in the mind of the public with substantiality. The usual building stones are inherently strong, but any stone that is sound structurally will have adequate mechanical strength for residential building.

Durability

The durability of a good natural stone likewise is usually more than ample to meet the probable term of useful occupancy. Good stone laid up with proper mortar should last for generations. Inferior material, which always should be avoided, includes badly weathered rock that might occur near the surface, greatly shattered or broken material, rock consisting of aggregates poorly cemented together, or rock too soft or too easily soluble to make good building stone. Sufficient time should be allowed, of course, between removal from the quarry and ultimate use so that the stone will be adequately seasoned.

Texture

Both fine-grained, homogeneous stone, and coarse-grained material are used in residential work. Choice is largely a matter of personal taste and judgment on the part of the buyer, governed by the availability of the stone.

Structure

Joints, bedding planes, or other seams or fractures in stone intended for residential work may be highly desirable or may entirely unfit the stone for structural use, depending upon their frequency and kind. The rock may be unsuitable for building purposes if undesirable seams or bedding planes occur closer together than the final dimensions of the building blocks. On the other hand, such structural characteristics are of advantage where the breaks occur more or less conformable to the size of the ultimate building unit. The planes of weakness or fracture then may be utilized in quarrying and shaping practice.

An example of a satisfactory building stone is a granite gneiss which has been so stressed that many joint planes have developed. Slabs and blocks of stone are readily pried loose from the quarry face with bars. Very light blasting is occasionally necessary. The face of the joints is attractively iron stained and these surfaces commonly are left for the exterior of the final masonry. As the stone is sound between these joint planes, the seams in this instance contribute greatly to the salability as well as the workability of the material.

With certain sandstones or quartzites, the natural bedding planes permit slabs to be split out readily. If the rock is too thinly bedded and the bedding planes are weak, the rock may be unfitted for structural purposes.

Massive rock, showing no seams or joints, may be more difficult and costly to shape. Massive limestone, or marble, however, yields readily to gang saws, Carborundum saws, diamond saws, or wire saws. Most true granite, even though it exhibits few or no seams or fractures can be split readily along the rift and grain.

Color.

Pleasing and distinctive color in a building stone is an important asset. Shades of gray, cream, buff, and light brown are popular in modern architecture. Other colors such as pink and red also are widely used where available. Warmth and life in color work seem among the first requirements, as dingy, drab, or dirty-looking material may be hopelessly unattractive. Color appeal is being capitalized by some quarry operators whose stone is distinctive and attractive.

DIMENSIONS OF RESIDENTIAL STONE

Sizes of stock offered for small-home construction vary somewhat to conform with the physical condition of the stone in the quarry. Quarrymen utilize natural seams where circumstances permit and the nature of the product is governed by such factors as kind and frequency of bedding planes, and kind and number of joint planes.

An examination of the size of quarry stock of a few large quarry operators who are active in the residential building field indicates the general types and sizes of building stone now on the market. Data here given include granite, limestone, and sandstone.

A quarrier of granite dimension stone offers to builders a rubble for ashlar work. For illustration, a block of granite from the quarry stock may be roughly 6 feet long, 2 feet wide, and 1 foot 6 inches thick. Such a block is split by shims and wedges in shallow drill holes into blocks 2 feet by 1 foot by 1 foot 6 inches. Blocks of this size are further split into pieces approximately 2 feet by 1 foot by 6 inches.

These slabs are ready for shipment to the building site. Two dimensions are established by the quarry operator. The width is the height of the stone as set in the final wall - in this case 1 foot. The thickness - 6 inches - remains unaltered. Lengths, however, are broken by the mason or his helper to fit the general wall style.

The dimensions of this rubble stock are random within the following limits; length 1 foot to 4 feet, rise 8 inches to 16 inches, thickness 5 inches to 8 inches. Such rubble strips need be broken but once or twice to provide building stone of final dimension. Included in the stock is a sufficient quantity of thicker pieces from which to make heavy returns, jambs, and other pieces of odd size.

The surface of these blocks is the broken texture of a hard granite which has been broken carefully along the rift and grain. Its relatively low selling price does not permit shaping or other manufacturing processes, but if desired, ashlar quarried to specific dimensions is provided. One ton of this one- and two-man rubble furnishes from 18 to 24 square feet of masonry veneer.

A producer of limestone for building purposes places upon the market material which has been sawed on four sides. These strips are broken to desired lengths upon the building site by the mason contractor. The sawed surface varies considerably in texture depending upon the process of sawing employed, different effects being secured by using various sizes of sand and steel shot.

The thickness of these strips is usually 4 inches, although material as thin as 3 inches sometimes is used where building codes permit a lighter type of construction. The length of the strips is random from 3 feet to 5 feet. The width of the strips corresponds to the height of the units desired. For usual work three unit heights are recommended, although four are sometimes used. The preferred unit heights are as follows: A, two units - 4 inches and $8\frac{1}{2}$ inches; B, three units - 4 inches, $8\frac{1}{2}$ inches, and 13 inches; C, four units - 5 inches, $7\frac{3}{4}$ inches, $10\frac{1}{2}$ inches, and $13\frac{1}{4}$ inches. Thus the thickness of the final stone in the wall is uniform, usually 4 inches, the height of the wall units corresponds to the width of the strip, and the length is determined on the job by the judgment of the mason contractor.

No lengths more than $3\frac{1}{2}$ times the height nor less than $1\frac{1}{2}$ times the height are recommended. Mortar joints are one half inch thick. Where a masonry backing is used, approximately 10 percent of the wall area consists of bonding stones 6 or 8 inches thick, tying the stone wall securely to the backing.

Dimensions of a well-known sandstone for residential work are as follows: Height 4 inches to 18 inches, thickness 4 inches (8 inches for bond stones), and lengths random but should be at least $1\frac{1}{2}$ times the height averaging about 24 inches. No lengths greater than 50 inches are recommended. These specifications are for ashlar sawn with sand, chat, or shot, depending upon the texture desired.

Specifications for split face material are as follows: Heights 2 inches to 10 inches, standard thickness 4 inches (8 inches bond stones), and length $1\frac{1}{2}$ times the height. The stock heights (or widths of strips as sold to the trade) are $2\frac{1}{2}$ inches, 4 inches, $5\frac{1}{2}$ inches, 7 inches, and $8\frac{1}{2}$ inches. One-inch heights are also available at a slightly higher price. One-half-inch mortar joints are recommended.

For small-house construction, heights of $2\frac{1}{2}$ inches, $5\frac{1}{2}$ inches, and 7 inches are recommended. By using these heights in split face ashlar work, a good random effect is achieved by making 12 percent of the wall in $2\frac{1}{2}$-inch heights, 46 percent in 4-inch, 26 percent in $5\frac{1}{2}$-inch, and 16 percent in 7-inch heights. Where an additional height unit of $8\frac{1}{2}$ inches is used, these

proportions may be modified so that the wall consists of 10 percent of $2\frac{1}{2}$-inch heights, 25 percent of $4\frac{1}{2}$-inch, 40 percent of $5\frac{1}{2}$-inch, 15 percent of 7-inch, and 10 percent of $8\frac{1}{2}$-inch heights.

Material is also available in the form of split face and sawed beds from which to make building units of standard brick dimensions (8 by $2\frac{1}{4}$ by $3\frac{3}{4}$ inches). This material is prepared in what one company calls brick strips $2\frac{1}{4}$ inches in height, $3\frac{3}{4}$ inches in width, and 6 inches to 36 inches in length.

Another sandstone company places building stone upon the market 4 inches to 15 inches in height, up to 20 inches in length, and 4 inches in thickness. It reccmmends that the length be about twice the height.

CONSTRUCTION METHODS

In modern construction natural stone is usually employed as a veneer, the several kinds of building stones requiring somewhat different practice which varies more in detail than in principle. Such variations are in reality a common-sense conformance to the physical properties and character of the particular stone.

Stone veneer for residences is applied as a facing over some form of backing which may be either frame construction, hollow tile, concrete, common brick, cinder blocks, Haydite, or other lightweight aggregate, or a combination of these or other building units.

Stone veneer over frame construction is probably the lowest-cost type of stone-faced wall. The stone veneer may be 4 inches or more in thickness where sawed stone or its equivalent is used, although exceptionally it is only 3 inches thick. For split stone such as granite, a 5- to 7-inch veneer is commonly recommended. A waterproof building paper or some similar material is placed between the wood sheathing of the frame construction and the masonry. Galvanized or rustproof metal anchors tie the masonry to the studding. The anchor heads are imbedded in the mortar joints or are bent down into niches hewn in the stone if the stone used is soft enough to be cut easily. Upon the interior faces of the studs any system of lath and plaster construction or its equivalent is used.

With backing of hollow tile, common brick, or other material, the stone veneer is bonded by rustproof metal anchors or by means of bond stones which firmly lock the veneer to the backing.

For best results the mortar is carefully prepared by experienced workmen and from selected materials. Proportions sometimes recommended are 2 parts lime, 1 part cement, and 9 parts of sand. The cement should be nonstaining and the sand should be well washed and well graded.

Construction details are variable and depend to some extent on building codes. The stone producer is usually guided by the advice of architects as to the most efficient way of utilizing the particular stone in question.

DEPRECIATION AND MAINTENANCE

Because of its permanence, stability, attractiveness, and low maintenance cost, natural stone is a highly favored material for residential as well as for other types of construction wherein depreciation and maintenance are important items to be considered. More and more stone is being used in medium-priced dwellings for which the initial outlay is no less important than the first cost plus maintenance over a term of years. Comparative figures are lacking for maintenance cost of stone as against other construction, but the saving with stone or other masonry is generally conceded. As the significance of depreciation as applied to dwellings is not generally realized, the following data are of interest (table 1):[3]

Table 1. - Probable useful life and depreciation rate of dwellings

	One-family dwelling		Two-, three-, or four-family dwelling	
	Probable useful life, years	Depreciation rate, percent	Probable useful, life, years	Depreciation rate, percent
Masonry, brick, concrete, reinforced concrete, brick and steel, steel frame, steel and stucco (fireproof)	50	2	45	$2\frac{1}{4}$
Masonry, slow-burning, with or without steel frame	50	2	40	$2\frac{1}{2}$
Masonry with frame interior	50	2	33	3
Frame	33	3	30	3-1/3

The difference between a depreciation rate of 2 percent and 3 percent obviously amounts to $50 a year on a $5,000 home, or 6 percent of $833, a sum which therefore might be a wise investment for more durable constructions. Possible savings in fire insurance premiums are also worthy of mention.

MARKETS

A popular, widely advertised stone may find buyers many hundred miles from the quarry. Ordinarily, however, the small producer or contractor is dependent upon local markets until his product achieves wider recognition. The most successful producers of residential stone have introduced their products gradually, fostering local outlets and allowing the demand to grow

3 U. S. Treasury Department, Depreciation Studies, Preliminary Report of the Bureau of Internal Revenue: January, 1931, p. 3.

as it will, slowly and normally, and encouraged by contacts which the quarry operator is able to make. Each house or other structure built of a distinctive building stone is a perpetual advertisement of the quarryman's product. The effect upon the prospective buyer is proportional to the number of such buildings, and to the artistic effects secured.

The market area of any given residential building stone is limited largely by the demand for the stone balanced against the increasing cost of the material, as transportation to greater distances from the quarry adds to the ultimate cost to the consumer.

Residential building normally accounts for one third to two fifths of the total value of construction contracts in the United States. The value of residential construction contracts alone in 1929 was about two billion dollars. Table 2 shows the relative importance of the several geographic districts in 1929. Data are partly from F. W. Dodge Corporation statistics.

Data regarding estimated building expenditures as ascertained from building permits in the larger cities are gathered currently by the United States Bureau of Labor Statistics. In 1928, for example, a survey in 310 cities with a total population of 44,940,049 indicated a per capita expenditure of $39.22 for housekeeping dwellings only. This figure varied from a minimum of $0.26 per capita in Butte, Mont., to a maximum of $243.64 per capita in Yonkers, N. Y.

During 1931 and 1932 a study was made by the Bureau of Labor Statistics regarding the relative cost of labor and material in building construction.[4] Data were gathered in 15 cities thought to be fairly representative of general conditions: Namely, Atlanta, Boston, Chicago, Dallas, Duluth, Indianapolis, Little Rock, New Orleans, New York, Roanoke, Saginaw, St. Louis, Salt Lake City, Seattle, and Trenton.

The investigation indicates that as an average for the 15 cities the residential-building dollar is spent as shown in table 3.

4 Bureau of Labor Statistics, "Relative Cost of Material and Labor in Building Construction 1931-1932." From the Monthly Labor Review, October, 1932, United States Department of Labor.

Table 2. - Residential contracts awarded in 1929, by territories[1]

District	Value residential contracts awarded (In millions and tenths of millions of dollars)	Percent of total
Metropolitan New York and vicinity	551.1	25.6
Chicago	315.4	14.6
Middle Atlantic	246.2	11.4
Western	239.5 (estimated)	11.1 (estimated)
Pittsburgh	166.1	7.7
New England	153.3	7.1
Southern Michigan ...	108.3	5.0
Southeastern	90.7	4.2
Kansas City	77.7	3.6
St. Louis	70.2	3.3
Texas	60.2	2.8
Up-State New York ...	41.4	2.0
New Orleans	17.9	0.8
Central Northwest ...	17.2	0.8
Total	2155.2	100.0

1 Metropolitan New York and vicinity includes northern New Jersey, New York City, Long Island, Westchester, Orange, Putnam, and Rockland Counties, N. Y.

Chicago includes northern Illinois, Indiana, Iowa, and eastern and southern Wisconsin.

Middle Atlantic includes eastern Pennsylvania, southern New Jersey, Maryland, Delaware, District of Columbia, and Virginia.

Western includes California, Washington, Oregon, Montana, Idaho, Wyoming, Nevada, Utah, Colorado, Arizona, and New Mexico.

Pittsburgh includes western Pennsylvania, West Virginia, Ohio, and Kentucky.

New England includes Maine, New Hampshire, Vermont, Massachusetts, Rhode Island, and Connecticut.

Southern Michigan consists of the southern peninsula.

Southeastern includes the Carolinas, Georgia, Florida, Alabama, and eastern Tennessee.

Kansas City includes western Missouri, Kansas, Oklahoma, and Nebraska.

St. Louis includes southern Illinois, eastern Missouri, northeast Arkansas, western Tennessee, and northwest Mississippi.

Texas includes the State only.

Up-State New York includes all counties north of Orange, Putnam, and Rockland.

New Orleans includes Louisiana, western and southern Arkansas, and eastern and southern Mississippi.

Central Northwest includes Minnesota, the Dakotas, northern peninsula of Michigan, and northwest Wisconsin.

Table 3. - Distribution of residential building costs by class of work and cost of material
(Average of 15 cities)

Class of work	Percent of total cost chargeable to specified class	Percent of total cost chargeable to material cost in specified classes
Excavating and grading	1.3	0.02
Brick work	14.8	8.6
Carpenter work (builders' hardware, lumber, and millwork)	27.3	18.3
Tile work	3.5	2.0
Concrete work	11.7	7.4
Electric wiring and fixtures	4.5	2.9
Heating and ventilating	6.6	5.3
Plumbing	10.1	7.6
Plastering	8.2	3.7
Painting	4.2	1.4
Papering	.5	.2
Roofing	1.8	1.2
Miscellaneous	5.5	4.1
Total	100.0	62.7

As indicated by this study, of every dollar that is spent in residential construction, 62.7 cents goes for material. Of this 62.7 cents 8.6 cents is spent for brick, 18.3 cents for lumber and related supplies, 2.0 cents for tile, and 7.4 cents for cement, sand, and gravel, or other aggregates or material pertaining to concrete work.

Expressed in general terms, the estimated expenditures in 1928 for building material to house one person amounted to about \$3.38 for brick, \$7.20 for lumber and related material, \$0.79 for tile, and \$2.90 for materials of which to make concrete.

An "average" American city of 100,000 inhabitants might therefore provide an annual market of \$338,000 for brick, \$720,000 for lumber and related material, \$79,000 for tile, and \$290,000 for concrete material, on the basis of the 1931-32 distribution study and of the 1928 building volume.

It is at all times dangerous to apply broad generalizations to specific cases, and it should not be assumed that any one city will provide a building-material market of the foregoing dimensions. Some communities absorb and use much more material per capita than others. The figures presented here are merely to indicate the possible extent of the market for residential building stone in the average community.

SALES PROBLEMS

Sales methods have been studied carefully by a number of quarry operators catering to the residential market, but no detailed account can be given here of the various means by which residential stone generally is sold. A few of the more common selling channels are mentioned, however, because the problems thereof differ somewhat from those pertaining to distribution of other building materials.

Commodities such as brick and lumber are more or less standard products stocked by most building supply dealers, but no uniformity of size, shape, or style prevails among the many building stones upon the market, nor are natural stone units stocked extensively by dealers. Much building stone is sold by the producer direct to the ultimate consumer with whom sales contacts are made direct or through architects and masonry contractors.

Only the very large and well-established quarry operator can afford to advertise nationally, and only a few stone districts can hope for a nation-wide market. The enormous expense of national publicity is justified only when a long campaign is contemplated, as immediate results may be somewhat intangible. For the small operator, advertising of this kind is out of the question. Some producers find it advisable, however, to stimulate public consciousness of their material by judicious advertising in local papers and trade journals. The publicity resulting from people's actually seeing a stone in structures that are popularly priced is considered one of the surest and cheapest forms of advertising.

One quarrier of an attractively colored building stone within a few miles of a city of a half million people has developed a market principally by creating an interest in structures in which his material is employed. His markets are local. His product was introduced slowly at first through the construction of several small buildings such as filling stations and residences. These jobs began to bring the stone to the attention of builders and the public. Each new job increased the demand for his stone and a profitable and expanding business has grown up with no great amount of selling expenses.

Quarry operators have found it advisable to establish definite contacts within the logical trade area for their stone. Certain producers acquaint architects with the different color effects that are possible, the sizes that are most economically available, the workability of the stone, and other facts relevent to the product. The architect's attention is often gained by personal contacts, by display of samples of typical color effects and textures, by invitation to visit the quarry, and by personal letters, pamphlets, or by general advertising constantly reminding him of the stone in question. It has been found that architects encourage the use of meritorious building stone because an attractive, well-built, and well-designed stone residence is an advertisement for the architect as well as for the stone itself.

Quarry operators not infrequently receive suggestions from the architect as to innovations and improvements in shape, size, or color selection of the building unit. The producer generally cooperates with the architect insofar as his ability permits, for such friendly relations and good will are of obvious benefit.

Residential stone producers also find it of advantage to foster friendly relations with mason contractors, and to see that the workmen are sufficiently well acquainted with the properties of the stone to use it in the most economical and desirable manner. Mason contractors, moreover, have valuable contacts with builders and contractors through which to further the sale of a particular stone. A cordial relationship with contractors may be built up by the same methods as have been suggested for contacting architects.

Some producers of residential building stone are prepared to furnish real assistance to customers after the stone leaves the quarry. Recommendations are often made to the mason or contractor as to setting the stone, the mortar to be used, and other precautions to insure long life of the masonry.

Details are important; for example, one stone operator strongly recommends that his product shall be pointed only with mortar that harmonizes well with the stone, usually just a shade or two darker in color than the tone of the stonework. Perhaps more operators will realize eventually that every job, large or small, represents a permanent advertisement of the quarry which will either help to sell more stone or will increase "sales resistance."

OTHER OUTLETS FOR STONE IN THE RESIDENTIAL FIELD

Quarry operators catering to the residential home field have found that other markets exist for their product in addition to the stone used only for wall construction. Certain natural stones are used for interior trim, fireplaces, flagstones, and for miscellaneous landscape effects such as sundials, bird baths, rock gardens, and aquariums.

Nearly any sound building stone may be used successfully in fireplace construction. One large company offers to builders a series of fireplace designs for which their stone is sawed to exact dimension so that the builder may purchase a complete stone fireplace which needs only to be assembled upon the job.

Fabrication of stone intended for such uses as fireplace construction suggests a considerable field for producers of stone that has unique color or other property of decorative value. Merchandising stone in this manner is merely applying factory methods to specialized products. It is possible that the idea could be used in a similar way for a variety of outdoor uses.

The various landscape effects that can be achieved by the use of natural stone furnish an attractive outlet for much stone that is otherwise unadaptable for standard types of construction. Such specialty markets do not

consume spectacular tonnages of material, but where such units are fabricated at the quarry and virtually "packaged" for the consumer the return would be much greater per ton of stone than for the more usual outlets.

It would seem, also, that this specialty market could be developed further by offering to the small-home owner a variety of units which he can assemble himself at relatively small expense and with a minimum of labor. Some of the largest magazines catering to country or suburban people have built up their enormous circulations chiefly on the policy of telling their subscribers "how to do it." This same principle might be turned to account by stone producers, who could package their products so that the buyer could assemble the stone pieces into their final form with a minimum of hard work.

Many types of stone products are adaptable as flagstones both for porches and for walkways. Sandstone and slate for example, are used for porches, walkways, pattern walks, stepping stones, terraces, and steps. Artistic effects are achieved by using multicolored stone flags set in irregular fashion. Incidentally the story is told of a man who imported an expensive Italian marble with which to construct a certain walkway upon his country estate. In the course of time the marble slabs were received and were carefully laid by the contractor in a regular and uniform fashion. The owner, however, was disgusted with the finished walk and immediately ordered it torn up. He then procured a heavy sledge hammer and proceeded to break the marble flags into all sorts of shapes, much to the bewilderment of the contractor. The irregular fragments were then relaid, producing a much more beautiful walk than before.

Slate is perhaps the only natural stone readily amenable to roof construction. Among the advantages of a slate roof may be mentioned exceptional durability and imperviousness to the weather. Certain slate roofs have been known to render adequate service for many hundred years. Also very attractive color patterns are possible. The harmony that exists between slate roof and stone exterior appeals to many architects and home owners.

No modern home is complete without adequate provision for heat and cold insulation, not only in the walls but also in the roof construction. Of the natural stones, pumice stone and certain kinds of massive diatomite have been used for insulation purposes. Pumice stone is used extensively in Italy for wall construction and to a lesser extent in a few of the western States. Diatomite has been used to a slight extent in California and one or two other western States as a sawed brick for wall construction. In powdered form it has been used as an insulation fill in walls and between joists in ceilings.